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Bulletin 70 Mineral Technology 2

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, Director

A PRELIMINARY REPORT
ON
URANIUM, RADIUM, AND VANADIUM

BY

RICHARD B. MOORE AND KARL L. KITHIL

WASHINGTON
GOVERNMENT PRINTING OFFICE
1913

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CONTENTS.

Page.
Preface.................... 7
Introduction.................... 9
Acknowledgments.................... 9
The carnotite deposits of Colorado and Utah.................... 9
  Description of deposits.................... 9
    Coal Creek, Colo.................... 10
    Skull Creek, Colo.................... 11
    Split Mountain, Utah.................... 13
    Green River, Utah.................... 13
      Content of Green River ores.................... 16
    Table Mountain, Utah.................... 16
    Thompsons district, Utah.................... 16
    Paradox Valley and surrounding districts.................... 18
      Long Park, Colo.................... 21
      Club Ranch, Colo.................... 24
      Saucer Basin, Colo.................... 25
      East Paradox Valley, north side.................... 26
      East Paradox Valley, south side.................... 26
      Bull Canyon, Colo.................... 28
      McIntyre district.................... 29
      Hydraulic, Colo.................... 29
  Origin of the deposits.................... 29
  Mining methods and cost of mining.................... 32
  Transportation and prices.................... 33
  Concentration of ores.................... 35
    Necessity for concentration.................... 35
    Wet concentration.................... 36
    Results of wet concentration tests.................... 37
    Dry concentration.................... 38
    Cost of concentration.................... 39
    Possibility of chemical treatment.................... 41
  General statement.................... 41
Pitchblende deposits.................... 43
  Description of deposits in the United States.................... 43
    North Carolina deposits.................... 43
    Colorado deposits.................... 43
      Kirk mine.................... 43
      German and Belcher mines.................... 45
      The Calhoun mine.................... 46
      The Wood mine.................... 46
      Waste of low-grade ore.................... 46
  Concentration of low-grade pitchblende ores.................... 47
  European pitchblende deposits.................... 47
  Uranium deposits in Portugal.................... 49
  Uranium ores in Australia.................... 50

Page.
Vanadium from ores other than carnotite 51
  San Miguel County, Colo., deposits 51
  Huerfano County, Colo., deposits 52
  Cutter, N. Mex., deposits 53
  Eagle County, Colo., deposits 53
  Deposits in other States 54
  Deposits of patronite in Peru 55
Production of uranium, vanadium, and radium 56
Uses of vanadium, uranium, and radium 57
  Uses of vanadium 57
  Uses of uranium 58
  Uses of radium 58
Radium institutes 61
Market value of radium salts 62
Radioactive methods for testing ores 63
  Method of using the electroscope for approximate determinations 64
  Method of using the electroscope for exact determinations 66
    Standardization of the electroscope 66
      Correction for standardization 68
    Method for exact determination of radium 68
Commercial methods of treatment of ores 69
  Bleeker process for the recovery of vanadium 70
  Haynes-Engle process for the recovery of uranium and vanadium 72
  Koenig process for the recovery of vanadium 73
  Fischer process of extracting vanadium from carnotite 73
  Method used by Primos Chemical Co 74
  Fleck method of extracting uranium and vanadium 74
  Radcliff method for complex radium ores 75
  Treatment of carnotite ores with nitric or hydrochloric acid 76
  Bleeker process of separating uranium from vanadium 78
  Process of extracting vanadic acid from copper vanadate 78
  Austrian method of extracting radium from pitchblende 79
Analytical methods for uranium and vanadium 82
  Methods for the analysis of carnotite 82
    Method for the determination of uranium and vanadium 82
    Method for the determination of vanadium 84
    Method for the determination of uranium 84
  Volumetric method for uranium in carnotite and vanadiferous uranium ores 85
  Gravimetric method for vanadium and uranium in carnotite and other ores 88
    Separation of alumina 89
    Rapid method for the determination of vanadium in ores in the presence of iron 90
Minerals of uranium and vanadium 91
Publications on mineral technology and methods of mining 96
Index 98

ILLUSTRATIONS.

Page.

PLATE I. Part of uranium and vanadium ore region in Utah and Colorado..... 10

II. *A*, Stratum of carnotite and vanadic sandstone of Thompsons deposits, Utah; *B*, Opening of North Star mine, Long Park, Colo., showing outcropping carnotite.................................. 16

III. *A*, Opening and dump of the Swindler mine, Long Park, Colo.; *B*, Cliff mine, Saucer Basin, Paradox Valley, showing 4-foot stratum of carnotite and vanadium ore between the points marked by the hat and the spade blade.................................. 24

IV. *A*, Cliff mine, Saucer Basin, Paradox Valley. Another view of 4-foot stratum of carnotite and vanadium ore; *B*, Electroscope for determining the radium content of an ore.............................. 26

FIGURE 1. Sketch map of Paradox Valley.................................. 22

2. Apparatus for separating emanation from uraninite................ 67

PREFACE.

Early in 1912, from information received by the Bureau of Mines, it became evident that large quantities of valuable material were being wasted in mining the rare-metal ores of the West. In pursuance of its endeavors to increase efficiency in the mining and treatment of mineral resources in the United States, the bureau assigned Dr. R. B. Moore, physical chemist, and K. L. Kithil, mineral technologist, to investigations covering the production of uranium and vanadium ores, the elimination of waste in mining, and the development of methods for working up valuable raw material into finished products. In the chemical side of the investigation Dr. Moore has been assisted by C. F. Whittemore, to whom credit should be given for the analytical work appearing in the following pages. Mr. Kithil has been engaged with Dr. Moore in a study of the general problem, and has personally given especial attention to the development of mining and concentration methods.

As a result of this investigation it has been definitely shown that, although the Austrian Government has conserved its own resources of uranium and radium by purchasing the Joachimsthal mines and by carefully supervising pitchblende production, the deposits of radium-bearing minerals in the United States are being rapidly depleted by wasteful exploitation, chiefly for the benefit of foreign markets.

Seemingly the country has been quite unaware of the extent to which uranium ores have been sent abroad. Investigation has developed the fact that during the year 1912 carnotite ores carrying 28.8 tons of uranium oxide were produced and that practically the entire amount was exported. The major part of this ore carried between 2 and 3 per cent U_3O_8, as it appears that no ore carrying less than 2 per cent can at present prices bear the cost of transportation. This means that 8.8 grams of radium chloride, or an equivalent 11.43 grams of radium bromide, will be obtained from the ore shipped from this country in 1912. Only one American company has been preparing radium salts of a high degree of radioactivity and its product has only recently been offered for sale. The American ores exported were purchased for their radium content. It is improbable that all of the ores exported are now represented by finished product, but the 1912 production of radium from American ores can not have been much less than the quantity mentioned, for uranium was being shipped abroad in almost equal quantities in 1911, and is still being sold for future manufacture. It can probably be definitely stated that in 1912 there was obtained from American ores nearly two and one-half times as much radium as from all other sources combined.

In the mining of uranium ore, for every ton marketed there are at present some 5 tons of low-grade material thrown on the dump and much more left in the mine awaiting only a feasible concentration process for commercial exploitation. Mr. Kithil has shown that elutriation can be applied to both uranium and vanadium ores and that a large proportion of the valuable material now going to waste can be readily obtained in marketable condition. If water is not available, air separators may be used, or other devices for dry separation. The uranium deposits of Colorado and Utah are being rapidly depleted for foreign exploitation, and it would seem to be almost a patriotic duty to develop an industry that will retain the radium in America.

Although no one can predict with certainty the value of radium or its possible application to science or medicine, the subject is certainly well worthy of investigation by our highest scientific authorities, and, under proper technical control, it opens to American industry a profitable field for exploitation.

Although at $90 per milligram for radium chloride the total value of the radium shipped in 1912 amounts only to $792,000 and the mining and separation of the ore can accordingly be considered only as one of our smaller industries, the fact should be noted that of this amount approximately $710,000 went into foreign hands and opened to foreign medicine and science opportunities in this most promising field that have been denied to our own people except by repurchasing the manufactured radium compounds at an almost prohibitive price.

France, Austria, England, and Germany have their radium institutes fostered by their Governments or by philanthropic foundations. Up to the present time, although the United States has been furnishing a large part of the material for these foreign investigations, comparatively little has been done in the hospitals and laboratories of this country.

Charles L. Parsons,
Chief, Division of Mineral Technology.

A PRELIMINARY REPORT ON URANIUM, RADIUM, AND VANADIUM.

By RICHARD B. MOORE and KARL L. KITHIL.

INTRODUCTION.

This bulletin presents a summary of available information regarding the sources of uranium, radium, and vanadium, the methods used in treating the ores, and the uses of the finished products. In particular the paper describes the ores found in the United States, giving especial attention to those characteristics of the ores and the conditions of their occurrence that affect mining and treatment.

ACKNOWLEDGMENTS.

In the preparation of this bulletin a number of persons rendered assistance by furnishing information and by making it possible for us to visit the carnotite deposits in western Colorado and eastern Utah at a time of the year when without this assistance it would have been difficult to obtain satisfactory results. We desire to express our thanks to the following persons: Henry Hall, W. L. Cummings, T. V. Curran, K. D. Hequembourg, David Taylor, H. C. Brown, Forbes Rickard, George W. Alsdorf, Dr. Herman Fleck, Orr J. Adams, C. S. Cherrington, Newt. Stewart, O. B. Willmarth, Angus Cameron, George A. Head, and Lorimer Bros.

In addition, we also wish to express our cordial thanks to Charles F. Whittemore for a number of chemical analyses made by him and for other valuable assistance.

THE CARNOTITE DEPOSITS OF COLORADO AND UTAH.

DESCRIPTION OF DEPOSITS.

Up to the present the carnotite deposits of Colorado and Utah have excited little interest outside of those two States, although they are the most important uranium-bearing deposits known in the world. During the fall of 1912 the authors were able to visit the more important localities, which are scattered over a considerable area, as shown by the accompanying map (Pl. I). A description of the deposits visited follows:

COAL CREEK, COLO.

The Coal Creek deposits of carnotite are 14 miles northeast of Meeker, the county seat of Rio Blanco County, Colo. The claims, 10 in number, are reached by wagon road and lie east of Spurlock Ranch, just beyond Henry Ranch. Little more than prospect work has been done and it is difficult to estimate the extent of the deposits.

The carnotite occurs in the lower bed of a group of beds of massive white sandstone that underlies the Dakota sandstone (conglomerate). The outcrops are on or near the hogback ridge that is formed by the lowest and most massive bed. For further details of the geology, the reader is referred to Gale's work.[a]

On the Caywood claims nine prospect pits were examined, of which several show no signs of carnotite. No. 1 shows none; in No. 2, about 1,000 feet above Coal Creek and 300 feet higher than No. 1, is an exposure of sandstone colored a light green in streaks and layers for a thickness of about 6 feet from the top; the entire rock is somewhat impregnated with the color, which is due to chromium. In this pit no carnotite ore is exposed. Some petrified wood was found at its bottom, also showing chromium stains.

The next pit, No. 3, just below No. 2, and southwest of it on the same hill, contains a petrified tree about 12 inches in diameter lying halfway across the pit. Cracks and interstices in this tree, which is rusty brown, are filled with powdery yellow carnotite. The sandstone for 1 foot below the tree carries good ore. Under the grass roots the sandstone is soft and rather heavily impregnated with carnotite. Farther below, in the white sandstone, a brownish-yellow streak occurs, underlying which is a somewhat richer yellow material a few inches in thickness. On the right side of the pit, near the bottom, is a brownish-colored rock containing vanadium.

About 15 feet northwest from pit No. 3 is another opening in which is exposed a part of the petrified tree that is exposed in pit No. 3. Interstices in the tree, as in pit No. 3, are filled with rich yellow carnotite. In this prospect hole no other signs of carnotite were observed. The next pit, No. 5, about 12 feet from the one just described, shows only some yellow-colored sandstone at its mouth. In the sandstone below is a layer of black to dark-brown, much decomposed, material, rich in vanadium.

East and about 1,000 feet higher up on the ridge, at an elevation of about 9,500 feet, and near the top of the highest peak of the mountain, are the so-called Elkhorn claims. There are two prospect pits, in the first of which, at a depth of 9 feet measured from the

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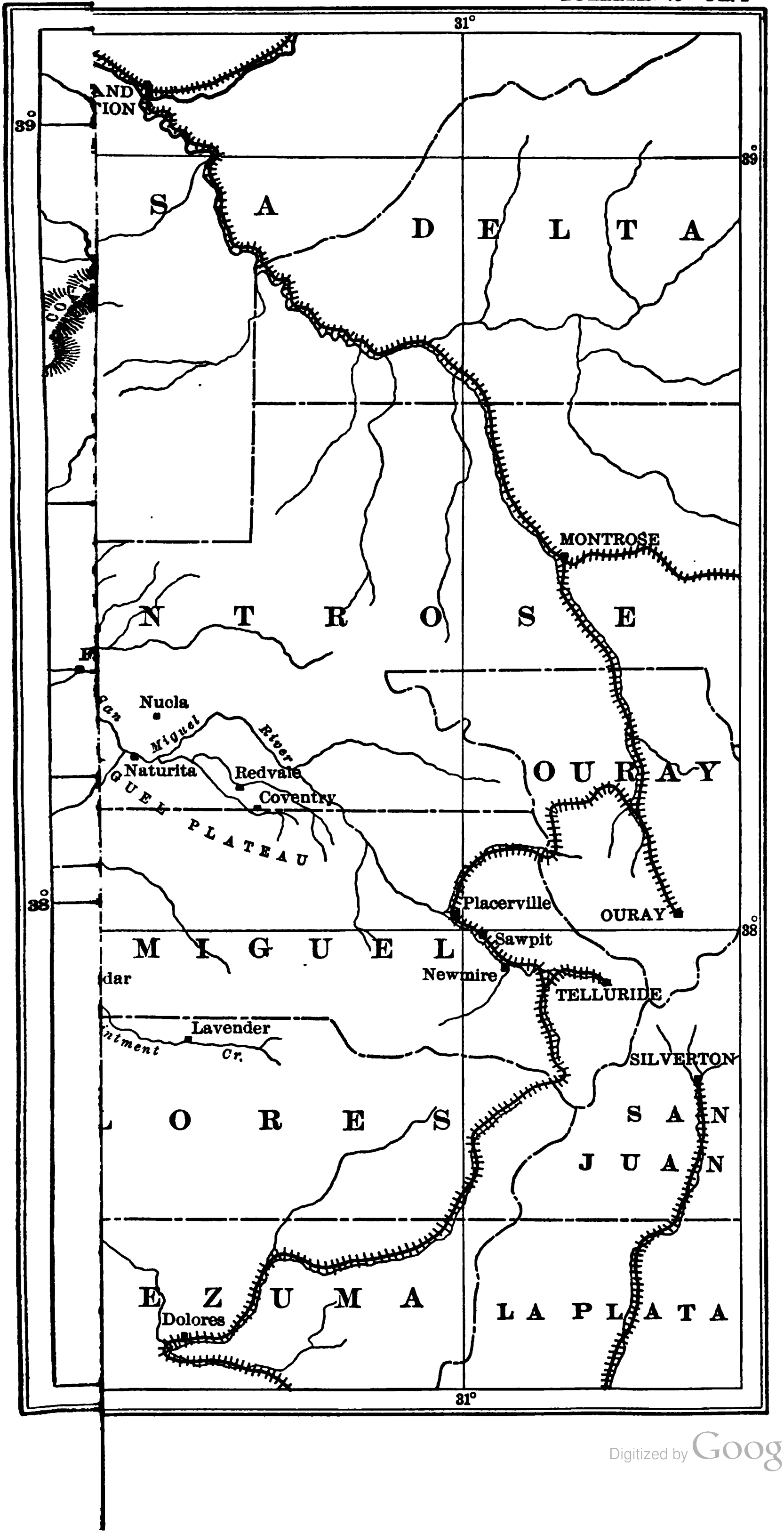
[a] Gale, H. S., Carnotite in Rio Blanco County, Colo.: U. S. Geol. Survey Bull. 315, 1907, pp. 110-117.


31°
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AND
TION
S A
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MONTROSE
Nuola
San Miguel River
Naturita
Redvale
Coventry
GUEL PLATEAU
O U R A Y
Placerville
OURAY
38°
38°
Sawpit
M I G U E L
Newmire
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ntiment Cr.
SILVERTON
L O R E S
S A N
J U A N
E Z U M A
Dolores
L A P L A T A
31°

top, a streak of dark-brown to black material about 12 inches thick is embedded in the sandstone that underlies the conglomerate (Dakota sandstone). No showing of carnotite was seen in the pit. The second opening, about 10 feet north of the first, shows the same occurrence of the dark-brown material near the top; about 10 feet below the surface is a layer of sandstone strongly impregnated with carnotite; this layer thickens to about 13 inches. The ore here is more like the carnotite-bearing sandstone of Paradox Valley in the southern part of the State. Scattered over the surface of the hillside are pieces of petrified wood.

On the west side of the same mountain, but below pits Nos. 4 and 5, is the Caywood No. 6 claim. Here the yellow color is noticeable only where the sandstone has been exposed to the air. Beneath the surface no stains are visible. The pieces of ore lying on the dump and exposed to the air show the discoloration to a marked degree. A bed of carbonaceous material 6 to 14 inches thick lies in the sandstone about 3 feet below the surface.

Several hundred feet to the southwest are the Caywood Nos. 7 and 8 claims. Pit No. 7 has no indication of ore. In No. 8 the sandstone is slightly discolored with yellow carnotite. Southeastward and down the same hill is a tunnel (No. 9) containing only thin streaks of yellow ore. At the bottom of the face of the tunnel is a streak of black carbonaceous material 2 inches thick.

The development at this place is not sufficient to give any definite idea of the extent of the deposits. They are patchy and it is far from certain that the ore will be found in commercial quantities.

The freight rate for ore is about 75 cents per 100 pounds from Meeker to Rifle, and about 25 to 50 cents per 100 pounds from the deposit to Meeker.

SKULL CREEK, COLO.

Another occurrence of carnotite and other vanadium ores is found in the northwestern part of Colorado. The deposits are in Skull Creek Basin in Routt County, 65 miles west of Meeker, 42 miles east of Jensen, Utah, and about 18 miles east of the Colorado-Utah line. The deposits are best reached from Mack, Colo., a station on the Denver & Rio Grande Railroad, by the Uintah Railroad (narrow gage) to Watson, and thence by stage to Vernal, Utah, where an outfit can be obtained to bring the traveler by way of Jensen to Skull Creek.

These deposits are between Wolf and Red Wash Creeks and are found in the foothills of the Blue Mountains, in the hogbacks of white sandstone.[a] Red clays and shales underlie the carnotite and vanadium bearing strata. Between the white sandstone which carries the carnotite and the conglomerate (Dakota sandstone) there is a series of beds composed of variegated clays and marls with limestone layers.

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[a] Gale, H. S., Carnotite and associated minerals in western Routt County, Colo.: U. S. Geol. Survey Bull. 340, 1908, p. 259.

Several of the claims were examined. The Lookout claim is on a hill, locally termed Uranium Hill, on the west side of Skull Creek Basin. The north side of the hill shows layers of green-stained white sandstone. The green material also occurs as incrustations and in many places is cemented with gypsum. The outcrops also show larger quantities of dark-brown and black materials, many of which carry vanadium. In two of the drifts the surface shows a beautiful color effect from green, dark-brown, black, and light-brown layers. The green color is due to malachite with many specks of blue azurite. These stains can be traced along the north side of the mountain toward the west for a considerable distance. Yellow coloring is rare.

Other claims are on top of Skull Creek Mountain, on the east side of Skull Creek Basin, about 500 feet above the valley. These deposits are difficult to reach as the steep ascent of an eroded crevice in the face of the rock on the south side of the mountain must be used. From the first pit in the Little Emma claim on top of the hogback 10 tons of carnotite was shipped to Germany, but the ore was found to be too low in U_3O_8. From all appearances it seems that the prospector has opened up the drift just above the carnotite-bearing horizon that lies under the excavations. The rock pitches about 40° SW. The sandstone that carries the carnotite also contains copper. Overlying the carnotite ore is much green-stained sandstone, its color being largely due to malachite. Underlying the carnotite is a thin streak of black, soft, coal-like material, and also an ore with green, as well as yellow, stains due to carnotite and probably malachite. The following layers were observed on the hanging wall of the opened drift, from top to bottom:

White material (calcite).

Sandstone with incrustation of bright greenish and yellow tint (1 to 2 inches).

Dark-brown vanadium and uranium ore with yellow specks (2 to 5 inches).

Thin sandy layer, dark brown to black (vanadium).

White sandstone with green impregnation (several feet).

Earthy blackish-brown streak carrying vanadium.

Several other claims lie along the hogback ridge for several miles in a westerly direction in the same group of strata. These were not examined owing to lack of time. The ores seem to be mostly vanadium and copper ores containing some uranium, the valuable minerals being in the white sandstone underlying the conglomerate frequently referred to as Dakota sandstone.

From the development work done it is impossible to obtain any correct idea as to the extent of these deposits.

SPLIT MOUNTAIN, UTAH.

A deposit of uranium ore had been reported at Little Split Mountain in the vicinity of Island Park, near the Horse Shoe Bend of Green River, Uinta County, Utah, but on investigation no such occurrence was found.

The road from Jensen by way of Rainbow Park to Island Park is rough, especially between Rainbow and Island Parks, across the mesa. The locality visited is about 30 miles northeast of Jensen and the trip takes an entire day. From Island Park a trail 6 miles long leads over steep mountains and through ravines to the claims on the north side of Green River. There are also some claims on the south side of the river, which cuts the mountain in two.

No uranium or vanadium ore was found. Only the claims on the north side of the river were investigated, there being several prospect pits, drifts, and shafts. Near the base of the mountain, just above the river, is a large mass of limestone breccia; overlying this is a decomposed shaly rock carrying iron but no carnotite. The layers are 20 feet thick and are thinly foliated. Just above the shale is a soft, brownish-black carbonaceous streak, 6 inches thick. Overlying this streak is decomposed material of shaly appearance, somewhat similar to that mentioned above. This is overlain by 3 feet of white sandstone that shows narrow veins of malachite and azurite and is well impregnated with these carbonates. Above the sandstone is 6 to 8 inches of black coal-like material. At the bottom of the drift is a black shale carrying streaks of white sandstone about 1 inch thick that show small round green and blue specks of the copper carbonates.

The south side of the river was not examined, but in the openings on the claims the same stratification is said to be visible. The ore-bearing bed continues on that side of the river, but 500 feet up the mountain side, showing that the river has cut it in two. There is said to be a 40-foot shaft and an open cut 75 feet long.

GREEN RIVER, UTAH.

West and south of Green River, Utah, the San Raphael swell reaches a height of several hundred feet above the surrounding plain. The main axis runs almost north and south. The eastern side is much eroded and the beds of sandstone, shale, and conglomerate are tilted at various angles. San Raphael River cuts eastward across the northern part of the swell, turns south through the valley that divides the "swells" from the "reefs," and finally again takes an eastward course through the "reefs" to join Green River.

The "reefs" dip at an average angle of about 30°. They are cut at right angles to the direction of the valley which divides them from the "swells" by a series of nearly parallel gulleys. Many of the gulleys are only 100 or 200 yards apart, although they cut the edges of the "reefs" for several miles. The conditions are clearly favorable for prospecting, the beds being freely exposed on both sides of the valley of San Raphael River and also in the gulleys. Little prospecting and practically no development has been attempted except in connection with ore bodies thus exposed.

Some of the vanadium and uranium deposits were visited by Boutwell [a] and later by Hess.[b]

Most of the carnotite deposits are 10 to 12 miles southwest of the town of Green River. There is a good wagon road with no steep grades, so that in hauling ore a round trip can be made in one day. On the railroad 6 miles from the mines is a siding, but several deep gulleys have to be crossed to reach it and the road is not good, hence the ore is preferably taken to Green River.

The majority of the deposits are exposed in the gulleys that traverse the "reefs." The carnotite is always in a rather coarse sandstone overlain with fine conglomerate. Much petrified wood is exposed, also bones and other fossils. As at Meeker, the carnotite stains are conspicuous about the wood, much of which is heavily impregnated. The yellow ore is found mostly in or near the wood or in cracks in the sandstone, although both the sandstone and the darker ores are lightly impregnated in many places. Yellow ore such as occurs in Paradox Valley is largely absent here, most of the ores being dark colored. They may be divided into four general types—the yellow carnotite, found mainly in cracks; a dark-brown siliceous ore impregnated with carnotite; a black ore much of which is associated with carbonaceous material and some of which carries stains of carnotite; and a grayish-brown laminated sandy shale, rich in vanadium and carrying some uranium. Parts of the deposits show mixed ores, but the majority of them can be classified as above.

Many samples that show little or no carnotite turn yellow on exposure to the air for several weeks. This yellowing takes place quickly if the ore is heated, as by laying it on the top of a stove. Such a test frequently shows uranium in ores not suspected of containing it, and as the test is well adapted to fieldwork, it merits a wider use.

The principal claims are the Lorimer and the Forsman. At the former is a good, permanent camp, and 8 to 12 men have been constantly employed during the year. Mining on the other claims has not been carried on systematically, ore being mined in small quantities from time to time and kept in Green River until enough for a car has accumulated.

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[a] Boutwell, J. M., Vanadium and uranium in southeastern Utah: U. S. Geol. Survey Bull. 260, 1905, p. 201.

[b] Hess, F. L., Carnotite near Green River, Utah: U. S. Geol. Survey Bull. 530, 1912, p. 18.

About a mile south of the Lorimer claims is the Little Ruth claim. Below 4 feet of fine conglomerate there is 3 to 5 feet of sandstone, then about 5 inches of dark vanadium ore that carries some carnotite. This can be traced through occasional outcrops for a distance of 25 yards. At places the sandstone is impregnated with carnotite. Below the pay ore is conglomerate.

North of the Little Ruth is the Napoleon claim, on which is a cut 8 to 10 feet long. A 2-foot layer of sandstone shows the color of ore, and there are some layers of black carbonaceous material. The ore as a whole is low grade.

South of the Little Ruth is the Vernon Junior claim. A cut shows conglomerate at the top, underlain by fine sandstone, 2 inches of black, rich-looking vanadium ore, and 15 inches of shaly sandstone, showing yellow stains of carnotite but evidently a low-grade ore. The sandstone above the black ore is lightly impregnated with carnotite.

Still farther south is the Wardvern claim, where ore has been taken out of a cut about 20 yards long. A 12-inch band of sandstone is impregnated with carnotite along the whole length of the cut. Above this sandstone band is a thin streak of black ore.

The Monayunk is one of the Lorimer claims. The ore body is exposed for about 40 feet, but it pinches out at the ends. At the thickest part color shows over a width of 3 feet. The sandstone has stains of yellow both above and below a streak of black ore which has some wood in it and appears to be partly carbonized.

On the Grand View claim is a cave about 50 feet long that extends across the ore-bearing zone. Black vanadium ore shows in patches and stains at places along the roof.

Some of the best ore at Green River is on the Melrose Discovery claim. The ore is exposed for a distance of 120 feet. Below the conglomerate is sandstone which overlies an ore body averaging 2 feet in width. Below the ore is conglomerate, sandstone, and a second ore body 8 inches thick.

Six miles south of the Lorimer claims are the Loveless-Forsman claims. These are situated in a less broken country at the foot of the "reefs," the ore outcropping at several places from under loose and soft sand. This sand carries carnotite at grass roots and is a low-grade ore.

About 3 miles north of these claims and on the west side of the river several tons of ore have been mined at the Morris claims, the only ones that have been worked on that side of the river. Most of the prospecting has seemingly been confined to the "reefs," little time having been given to looking for ore along the "swells."

CONTENT OF GREEN RIVER ORES.

Shippers from Green River have had trouble in connection with analyses of their ores. Analyses of six lots of ore, a total of 36,100 pounds, showed an average content of 2.53 per cent V_2O_5 and 3.33 per cent U_3O_8. These results, however, were questioned, and the ore was delayed in transit for some time pending a settlement. There is no doubt that little, if any, ore shipped out of Green River has assayed over 2 per cent U_3O_8. Most of the ore carries between 1 and 2 per cent. The vanadium content in the above analyses is probably correct. Some shipments have assayed as high as 8 per cent V_2O_5. Owing to the ore being low in uranium, it must be picked carefully in order to get ore of shipping grade, 2 per cent. Picking causes considerable waste; moreover, efficient picking is difficult, owing to the character of the ore, many of the dark samples that seemingly show little yellow carnotite being richer in uranium than some highly colored samples.

The operators at Green River have the advantage of a lower combined haulage and freight rate than those at any other locality producing carnotite. The transportation charges from the Lorimer camp to New York are $17.50 per ton in carload lots. The only other locality having an approximately similar rate is Thompsons, Utah. It costs more than $17.50 to haul ore from any point in Paradox Valley to the railway at Placerville, Colo.

During the year 1912 346 tons of ore was shipped from Green River, Thompsons, and Cisco, the only points in Utah that shipped ore. A part of this carried less than 1 per cent U_3O_8 and the returns did not equal expenses. It is doubtful whether any of the ore carried 2 per cent U_3O_8; most of it carried about 1.5 per cent. Approximately 125 tons additional was mined and stored but not shipped.

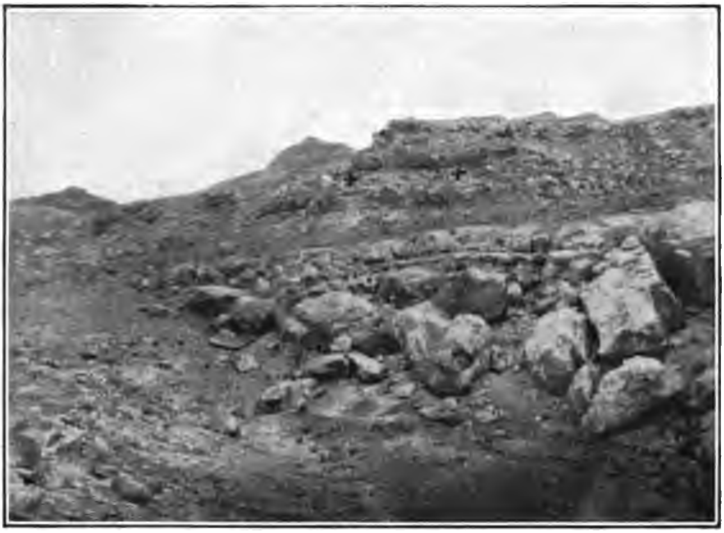
TABLE MOUNTAIN, UTAH.

Forty-five miles south of Green River is Table Mountain, where are deposits of red calcium vanadate, and carnotite in beds of sandstone several feet thick carrying both uranium and vanadium. Lumps of asphaltic material containing uranium and vanadium are frequently found.

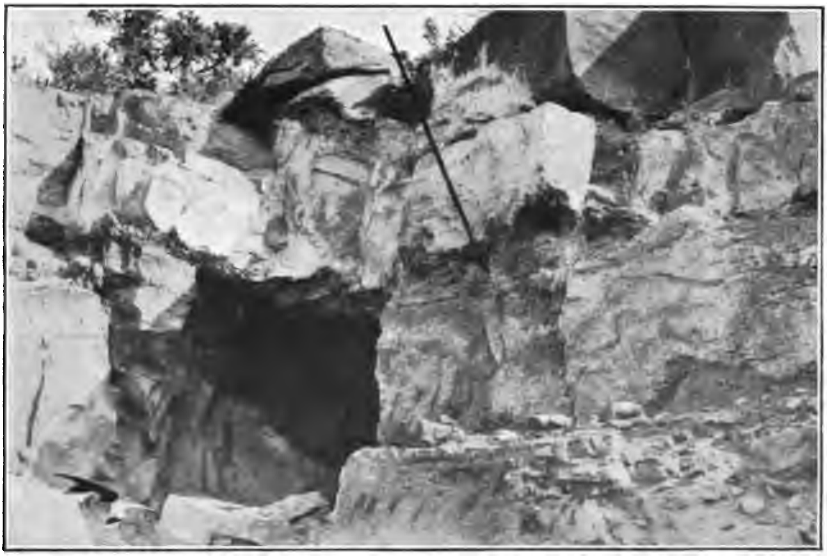
THOMPSONS DISTRICT, UTAH.

The Vanadium Ores Mining & Milling Co. claims are about 16 miles southeast of Thompsons, a station on the Denver & Rio Grande Railroad, in Grand County, Utah.

The deposits (Pl. II, *A*) are best reached from Thompsons over a fairly level road that after having traversed the surrounding foothills crosses a long stretch of desert lands in a southeasterly direction.



A. STRATUM OF CARNOTITE AND VANADIC SANDSTONE (SEE CROSSES) OF THOMPSONS DEPOSITS, 16 MILES SOUTHEAST OF THOMPSONS, UTAH.



B. OPENING OF NORTH STAR MINE, LONG PARK, COLO., SHOWING OUTCROPPING CARNOTITE.

About 13 miles from Thompsons the desert changes into a rugged hilly country. The road leads through small winding canyons to the camp, which is erected in an opening of a canyon near a spring.

The deposits are 1½ miles north and west of the camp and cover a considerable area.

During the night of our arrival in camp heavy snow fell, making a thorough examination of the claims impossible except where the surface of diggings and outcrops was exposed.

There was an exposed outcrop on Telluride No. 8 claim along the perpendicular face of a hillside. The yellow impregnations in the sandstone can be followed along the side of the hill for a considerable distance. A quantity of low-grade ore is lying on the dump. The carnotite-bearing zone of the sandstone is at this place 8 to 9 inches thick. Overlying this is a strip of carbonaceous material. Two feet below the carnotite there is a layer of what looks to be roscoelite 2 feet thick.

At different places in the sandstone are small pockets of red vanadium ore, probably calcium vanadate. This ore shows imprints of fossil-fern leaves and stems.

Some richer carnotite was found on claim No. 5, where at various points is a grayish rock, the cracks and fissures of which are filled with incrustations of rich yellow mineral. Some ore at this place is also found in pockets. In the same deposit there is a black shaly material the surface of which, after exposure to the air, turns decidedly yellow.

A decomposed sandstone is found in streaks, small layers, and pockets; it is greenish-yellow and carries both vanadium and uranium in small quantities.

Claim No. 1 shows at several places a thin streak of sandstone impregnated with carnotite. There are small pockets of a mixture of ores in which the carnotite, as well as the dark vanadium ore, is embedded in and between thin layers of crystallized gypsum. The little of this ore in sight is high grade.

Near the spring on claim No. 1 and near the top of the hill there is a small quantity of coal-like material in a hole about 3 by 10 inches. This material is highly radioactive, and after having all of the adhering carnotite removed by washing and crushing it is quite as radioactive as before. It may therefore carry uranium oxide.

On top of the hill at various points in the same canyon there are several outcrops of uranium and vanadium ores. These ores are embedded in a greenish-colored sandstone which constitutes a considerable part of the rock on the hillside. At one locality, at a distance of 1 to 3 feet above the lower stratum of the sandstone, the yellow carnotite is found in a loose sand. It is of low grade, but might make a milling ore.

The deposits form a connecting link between those of the San Raphael swell on the west and those at Richardson on the southeast and, with several breaks, ultimately reach the districts near Hydraulic, Club Ranch, Saucer Basin, and Long Park.

The Thompsons deposits are almost flat-bedded, but there is an unconformity and some faulting. At the time of the visit no work was being done, but during 1912 some ore was shipped, which, however, was held up during transit, as it contained less than 2 per cent of uranium oxide. The vanadium content was relatively high. This trouble can be obviated by more careful sorting. The ores in this district as a whole are mostly of low grade, but can probably be concentrated. It is difficult to estimate the possible amount available. Mining has been confined to the best outcrops, and the low-grade ore from hand sorting is thrown on the dump.

The water supply at or near the mines in this district is limited, and timber has to be brought from a distance. Grand River, near its confluence with the Dolores, is 8 miles west of the camp.

Other deposits of uranium ore in Utah are at Richardson, Fruita, and Moab. From Moab some low-grade material has been shipped.

PARADOX VALLEY AND SURROUNDING DISTRICTS.

The carnotite deposits in San Miguel and Montrose Counties, Colo., have been known for a number of years. As far back as 1881 Andrew J. Talbert mined some ore and sent it to Leadville, where it was tested for gold, silver, and copper. The report stated that it carried $5 gold per ton. In 1896 Gordon Kimball and Thomas Logan sent specimens to the Smithsonian Institution at Washington, D. C., and were informed that the mineral contained uranium. Shortly afterwards Kimball and Logan mined 10 tons of ore and shipped it to Denver where it was sold for $2,700. In 1899 Poulot and Voilleque, two Frenchmen, visited Paradox Valley, collected specimens, and sent them to Friedel and Cumenge in France, who announced in the French journals the existence of a new mineral which they named carnotite and described as potassium urano-vanadate.

Poulot and Voilleque in 1900 began operating at a copper mine at Cashin in Paradox Valley, where they used leaching vats to extract the uranium. Shortly afterwards they built a small mill in the McIntyre district, south of Paradox Valley. In this project they had the cooperation of James McBride of Burton, Mich. The mill ran until 1902, and during this time produced about 15,000 pounds of uranium oxide. The mill was started again in 1903 by the Western Refining Co., but ran only until 1904. Shortly afterwards the Dolores Refining Co. built a new mill a short distance from the old one, but after running for some years it, too, shut down. The concentrate, which was obtained by the Engle process, retained uranium and vanadium only, not the radium. In addition to this concentrate, some ore was shipped during this period. In 1912 the American Rare Metals Co. acquired the mill of the Dolores Refining Co. and is now operating.

Although the McIntyre district was the scene of some activity, little was done in the Paradox district until the formation of the General Vanadium Co. in 1909. That company began work early in 1910, in the same year that the Standard Chemical Co., of Pittsburgh, Pa., entered the field, and these two companies are now the largest operators in the district. The former have at present about 60 claims and the latter 90. The total number of claims filed is between 500 and 600, but a number of these are practically worthless. With reviving interest in the deposits claims were rapidly located, and during the last two years the output of ore has been the largest in the history of the industry.

Hillebrand and Ransome[a] have written concerning these deposits. Fleck and Haldane[b] have given a very complete account of the deposits in the Paradox and surrounding districts during the early stages of the work. Hess[c] has written a short account of some of the claims. In addition, a number of articles have been published by the mining journals. However, the field of operations has, to a large extent, changed since the articles mentioned were written, old deposits having been worked out and new ones opened.

The Paradox Valley is at the western end of the high plateau that slopes westward from Norwood to the eastern base of the La Sal Mountains. The Dolores River enters on the south and runs across the valley to the northeast, instead of following the valley lengthwise along the natural grade. Hence the name "Paradox."

The deposits are confined to a well-defined area. The eastern boundary can be represented by a line drawn from a point a little east of the junction of Dolores River and Disappointment Creek on the south, through a point 6 miles west of Naturita, and thence due north to San Miguel River. The western boundary is the La Sal Mountains which extend north beyond Uranium almost to Gateway. The total length of this area from north to south is about 40 miles and the width 20 miles. On the western side of the La Sal Range, in Utah, are the Moab deposits, and north of them are those of Richardson and Thompsons. Farther west are Green River, Table Mountain, Pahreah, and the other Utah uranium fields. If it were not for the break caused by the La Sal Mountains, a close connection might be drawn between the Utah and Colorado deposits. The main difference is that the former are of lower grade and more widely scattered.

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[a] Hillebrand, W. F., and Ransome, F. L., Carnotite and associated minerals in western Colorado: Am. Jour. Sci., vol. 10, p. 134.

[b] Fleck, Herman, and Haldane, W. G., A study of the uranium and vanadium belts of southern Colorado: Rept. State Bureau of Mines, Colo., 1905-6, pp. 47-115.

[c] Hess, F. L., Carnotite near Green River, Utah: U. S. Geol. Survey Bull. 530, 1911, pp. 161-164.

The principal localities where work has been done during the last two years are Long Park, Club Ranch, Saucer Basin, Hydraulic, north and south sides of East Paradox, Bull Canyon, and the McIntyre district. Roc Creek, the west end of Paradox Valley, and other localities have also furnished some ore.

The ores of the Paradox district differ in many respects from those of Utah, chiefly in carrying larger proportions of carnotite, and, as a rule, are more yellow. Not only do they carry more uranium, but also on the average more vanadium, although individual shipments from Utah might seem higher in vanadium than the average from the Paradox district. The following analyses of samples from several shipments of ore from Long Park during the past year will give some idea of the kind of ore that is being handled:

Analyses of ores from the Paradox district.

Constituent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
U_3O_8	3.14	2.83	3.35	2.75	2.48	2.91	3.43	3.54	2.95
V_2O_5	5.33	5.14	5.81	5.08	5.09	4.25	13.63	6.66	5.93

These analyses show a vanadium and uranium content higher than the average, a considerable part of the ore shipped containing as low as 2 per cent U_3O_8, whereas a large amount of lower-grade material is left in the mine or thrown on the dump. Occasionally a few hundred pounds of high grade ore carrying 15 or 20 per cent U_3O_8 is obtained from "bug holes."[a] All the shipments from Paradox Valley and the surrounding districts during the past year would probably average about 2½ per cent U_3O_8 and between 3 and 4 per cent V_2O_5.

The most typical ore is a sandstone so impregnated with yellow carnotite that the color is decidedly noticeable and containing small kidneys of brown sandy clay. The kidneys constitute a considerable part of some of the ore; in many cases they are thinly scattered through the sandstone. It seems to be generally acepted among the operators that the kidneys are rich in vanadium. The samples we have tested show vanadium. Although ore of the character mentioned is widely distributed in the Paradox and adjacent districts and constitutes a large part of the ore shipped, it is by no means the only ore of commercial importance. Indeed the variety of the types of ore here and also in Utah is one of the interesting features of the uranium and vanadium deposits. We have had time to test only a few of the large number of samples taken. There are dark-blue, brown, and black vanadium ores, the dark-blue ores being lustrous when first mined and usually carrying uranium; high-grade carnotite in "bug holes," so soft that it can be molded in the fingers; the same kind of ore crystallized with gypsum; and red calcium vanadate, some in radiated form, and some mixed with carnotite and blue vanadium ore. Much of the very low-grade ore on exposure to the air weathers to a green, rose, or yellow color, or to all three colors intermingled. In many places several ores of different types are mixed in an intricate mass; in other places the sandstone is impregnated along the lines of stratification and there are alternate layers of carnotite and dark vanadium ore. It can readily be seen that it requires considerable skill and experience to sort such ores properly, especially as the sorting is done on the basis of the uranium content and not the vanadium content.

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[a] A local term meaning a small pocket, lined with rich ore, in the formation. It is probably derived from "vug."

The deposits are invariably pockets, many of which, however, are of considerable size; 50 tons of shipping ore from a single claim is not unusual. Several claims have yielded more than this. Many of the pockets are exposed in the sides of the canyons, but at other places, notably Long Park, development work has to be done. The ore is found in a light-colored sandstone overlain in places with shale and conglomerates. According to Hillebrand and Ransome this is the McElmo formation.[a] The conglomerate is seen especially well in crossing from Club Ranch to the Saucer Basin. Below the McElmo formation lies a fine-grained sandstone (La Plata), and below this the Dolores or Red Beds.

LONG PARK, COLO.

The distance from Placerville, Colo., the nearest railroad point, to Long Park is 56 miles (fig. 1), which can be divided as follows: Forty miles to Naturita, the terminus of the stage route, 4 miles to Coke Ovens, where the Standard Chemical Co. has its headquarters and a large ore house, and thence 12 miles to Long Park. Coke Ovens is at the head of Paradox Valley, and the road from that point gradually climbs the escarpment on the north side of the valley to a height of about 1,000 feet, then drops 200 feet on the other side into Long Park, which is 6,500 feet above sea level. The park is about 3 miles long and one-half mile wide. The surface is gently undulating, with a gradual rise on the south side to about 200 feet, the north side being more precipitous.

The southern side of the western end of the park is traversed by two shallow, rather narrow valleys running from west to east. In the first of these are the Turner claims. At the time of our visit a new pocket had just been exposed. The ore was found at a depth of

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[a] Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, p. 11.

2 feet in a shallow cut in the side of the spur between the two valleys. The carnotite-bearing streak in the sandstone was about 8 inches wide where first cut; it thinned to 2 inches and then widened again to about 8 inches. There is considerable gypsum present.

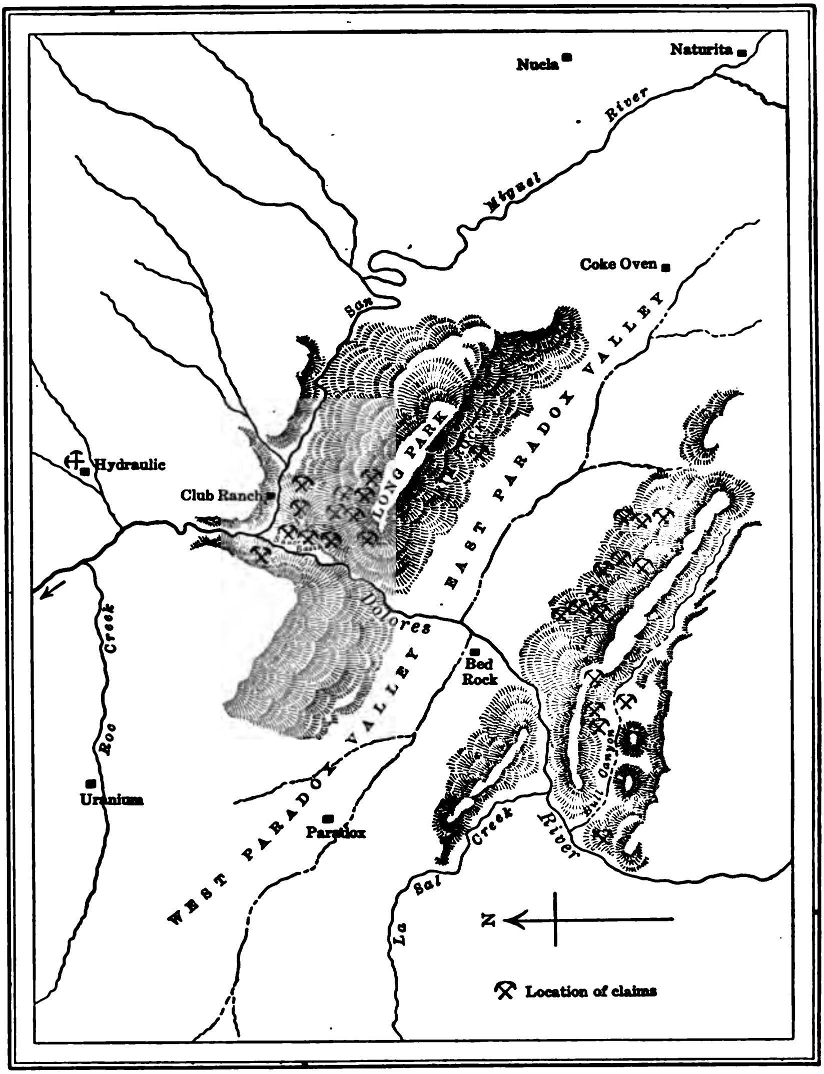


FIGURE 1. Sketch map of Paradox Valley.

On the south side of the same valley and opposite the newly found pocket is a large cut that has furnished much shipping ore. The dip of the beds is about 20° S. The ore remaining is both yellow and black in color, the average being a grade that would require careful hand sorting. There is a large quantity of milling ore on the dump.

Farther south is the Turner No. 4 claim, which shows 5 or 6 feet of massive sandstone underlain with 3 or 4 feet of hard shale partly impregnated with carnotite and containing small pockets of richer ore. Some dark vanadium ore is also present. The underlying sandstone is gray.

Several other openings on this property have yielded shipping ore, but were not being worked at the time of our visit.

In the second valley are the Curran claims, 13 in number. They are mostly situated along the north side of the lower half of the valley. Several promising openings are in the face of the low cliff overhanging the valley (Pl. II, B). Both tunnels and drifts are being driven and also open cuts. On the Cripple Creek claim the ore appears to have collected in a V-shaped pocket, the country rock being white sandstone. At the top of the pocket there are 15 inches of a dark-gray vanadium ore, and below this 4 feet of good uranium ore consisting of carnotite mingled with a bluish-black vanadium mineral, which lies in the carnotite in small local masses and stringers. The pocket dips in the direction of an ore-bearing zone observed in a short adit a little to the east and at a lower level, indicating that one may be a continuation of the other.

The Swindler claim (Pl. III, *A*) is to the west of the Cripple Creek. Here nearly 200 feet of tunnel work has been done, the ore having been followed from the outcrop on the face of the cliff. The ore is very similar to that of the Cripple Creek, the width varying at different points. On the dump is considerable low-grade material which represents the waste after the ore has been hand picked. Vanadium ore low in uranium is left in the mine. A number of sacks of rich ore has been obtained from "bug holes," or rich pockets.

At the upper end of the same valley is the camp of the Radium Extraction Co. which, with the General Vanadium Co., is a subsidiary company of the International Vanadium Co. of Liverpool, England. The latter is closely associated with the George Blackwell Sons Co. of Liverpool.

The claims of the Radium Extraction Co. are situated on the hogback north of the camp. There are three drifts; two of them are driven northwest and seem to be in the same body of ore. Lower on the hillside another drift has been driven to cut the ore body which seemed to strike in that direction. Over the ore is a thin bed of sandstone colored red by numerous small spots of iron oxide. Below it is 8 inches of medium-grade ore, then 1 to 2 inches of high-grade ore, and below this 2 feet more of medium ore. The ore dips 10° N. and strikes 10° E. Underlying it is a dark sandstone probably carrying a little vanadium, but this is not being mined. The ore was discovered 12 feet in the drift, from which point the ore body broadened to the dimensions stated.

In drifting for ore the miners follow the upper surface of the gray sandstone, and the appearance of splotches of dark-red and almost black ore is taken as an indication that carnotite is near.

In the lower part of the carnotite-bearing strata are black patches of what appears to be good vanadium ore. This is thrown on the dump.

The ore mined is hand sorted into three grades, high, middle, and low, in an average proportion of 3 sacks of high to 15 of middle and 23 of low grade.

The beds show an unconformity and some are faulted. Mining and development are carried on at the same time.

The Crucible Steel Co. claims, 7 in number, are also in Long Park. Most of them are near the Radium Extraction Co. claims, on the same hill. All assessment work has been completed, but no mining for shipping purposes has been carried on. Most of the prospect holes show good values. In ore in a ravine about 200 yards southeast of the Radium Extraction Co. claims are several cylindrical sleevelike pockets, or "bug holes." They are 2 to 4 inches in diameter and are filled with canary-yellow high-grade carnotite. The ore when first removed is so soft that it can be molded in the hand, but on exposure to the air it becomes hard and brittle. At these pockets the ore is surrounded by a hard envelope of ferruginous sandstone about 2 inches thick, outside of which the sandstone is soft for some distance. Such a hard coating is not found at all "bug holes," the ore at some being surrounded by soft sandstone. Occasionally the envelope is largely gypsum, showing that calcium sulphate was carried by the original mineral-bearing solution and crystallized out first.

In several places the ore-bearing sandstone contains carbonaceous material that has a yellow coating. This carbonaceous material is radioactive, but contains no vanadium.

There is another good showing of ore at an outcrop in a ledge about 75 feet above the spring close to the camp of the Radium Extraction Co.

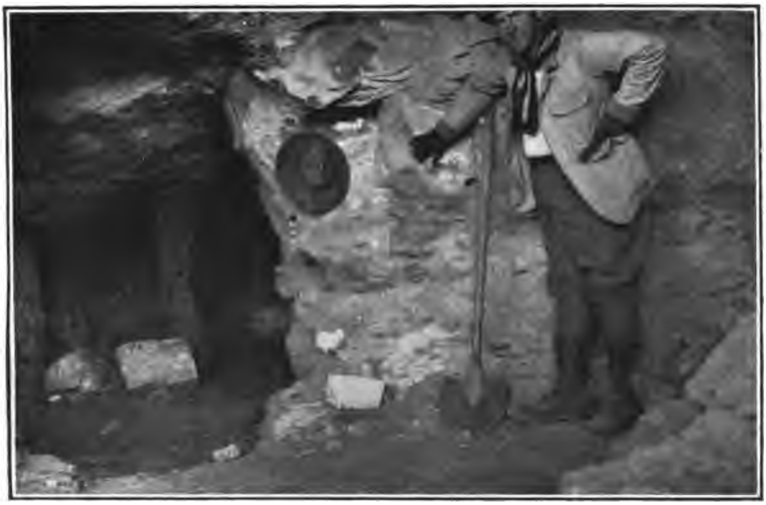
The Primos Chemical Co. claims, 7 in number, are in and around Long Park. Most of them are along the edge of a cliff, 3 miles north of the camp of Curran and the Radium Extraction Co. Assessment work only is being done. Little ore is exposed and this is rather low-grade vanadium ore.

In addition in and near Long Park are the Standard Chemical Co., Patton, Furr, Stone, and Rossler claims.

CLUB RANCH, COLO.

The Club Ranch deposits are about 7 miles from Long Park. There being no road between the two places, it is necessary to follow a rather difficult trail across rough country. The ore must be packed

A. OPENING AND DUMP (BELOW ARROW) OF THE SWINDLER MINE, LONG PARK, COLO.



B. CLIFF MINE, SAUCER BASIN, PARADOX VALLEY, SHOWING 4-FOOT STRATUM OF CARNOTITE AND VANADIUM ORE BETWEEN THE POINTS MARKED BY THE HAT AND THE SPADE BLADE.

from the Club Ranch claims on burros to Long Park, from which point it is sent in wagons to Placerville. The main operator at Club Ranch is the Standard Chemical Co., which has an ore house at Long Park for convenient transfer of ore from burro to wagon. The deposits can also be reached from Nucla. A road passes near Club Ranch, where, after fording the river, one can take a steep trail up the cliff on the south side of San Miguel River.

There are a number of open pits, from some of which considerable ore has been taken. Several tons of low-grade material lie on the dump, much of which probably carries 1 per cent U_3O_8. The ore seems to strike south and dip east. The overburden is usually 3 to 4 feet thick. The overlying sandstone, as well as the sandstone that carries the ore, contains thin seams of crystalline gypsum. Considerable work has been done here and it was stated that over 900 sacks of ore had been mined and packed from the claims. Nothing was being done, however, at the time of our visit.

The formation seems to be uniform throughout. The ore is in more regular layers than in the Long Park deposits, and there is more blue, black, and gray vanadium ore and less carnotite, the latter lying between layers of vanadium ore. In many respects the deposits are similar to those at Thompsons and San Raphael swell.

The overlying sandstone is filled with small, round, reddish-brown specks of iron oxide, similar to that in Long Park.

SAUCER BASIN, COLO.

Three miles south of the camp of the Standard Chemical Co. at Club Ranch are the four Wilson claims, which constitute the Cliff mine. The mine is in the rim rock and lies 500 feet above the floor of Saucer Basin and 6,100 feet above sea level. To the west, at a distance of 1½ miles, is the Dolores River, which cuts across the canyon at its western end after having traversed Paradox Valley.

A layer of carnotite-bearing sandstone extends 500 feet along the face of the cliff (Pls. III, *B*, and IV, *A*). It has more the appearance of a vein than any other similar deposit in the Paradox district. The pocket of ore is almost horizontal at the surface, but at 15 feet in, it begins to dip slightly to the north. At the top is white sandstone, then 4 to 6 inches of carnotite ore showing thin dark-gray and black layers of vanadium ore. Below this is 3 feet of similar vanadium ore that carries carnotite and in places narrow bands of decomposed quartzite. There seem to be two ore bodies with 3 feet of barren sandstone between; perhaps there is only one body, which is faulted. In places the barren sandstone carried a little carnotite, the result of leaching.

Most of the material exposed is low grade, but it has been found necessary to work one claim only thus far, owing to the fact that the drift has been in pay ore nearly all the time. This claim has been one of the heaviest producers in the history of the industry. Some prospect work is now being done on the property.

On a ledge a short distance from the mine a forge had been in use for some time. The heat from the fire caused some of the surface of the near-by sandstone, previously uncolored, to show a distinct stain of carnotite.

At various times considerable high-grade ore has been obtained from "bug holes." A quantity of low-grade milling ore lies on the dump, but seems to be mixed with waste rock. A rather large quantity of low-grade ore has been left in the mine. At one place the deposit widened to 14 feet.

The overlying sandstone makes a good roof and little timbering has to be done. Pillars of ore are left in the workings, or, where rich ore is encountered, pillars are built of waste rock.

By building an aerial railway from the mine openings over a low hill to the west the ore could be carried to the banks of Dolores River, where a small mill could be erected for concentrating it, or the ore might be treated nearer the mine by piping water from the river. At present the ore is packed on burros 10 miles, by way of Club Ranch camp, to Long Park, thence hauled in wagons to Placerville.

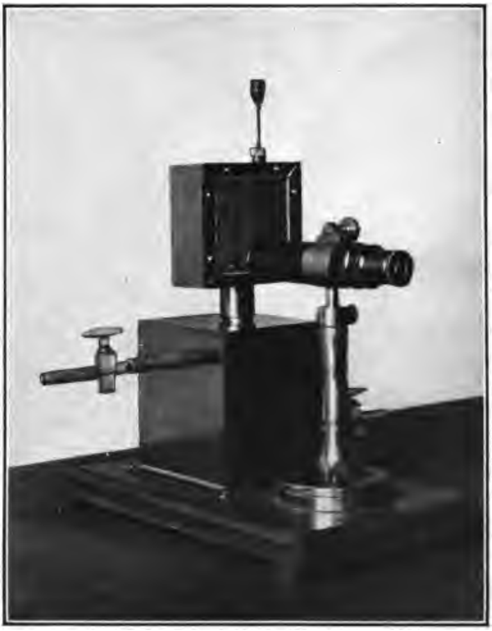
EAST PARADOX VALLEY, NORTH SIDE.

Two hundred feet above the road leading from Coke Ovens to Long Park and about halfway between these two places are the Jacobs and McKeever claims, on which are two prospect cuts and two or three short tunnels. The variety of ores in a small space is larger here than at any other locality in the Paradox district—carnotite; calcium vanadate; carbonaceous material that is highly radioactive and is also rich in vanadium; blue, black, and gray ores of vanadium; and "bug holes" containing carnotite. The ores are intermingled rather than deposited in layers. Although the ore body is 2 to 3 feet thick, it is difficult to estimate the extent of the deposits, as not enough development work has been done. A recent shipment of 13 tons from this place carried 3.43 per cent U_3O_8 and 13.66 per cent V_2O_5.

EAST PARADOX VALLEY, SOUTH SIDE.

Along the road into Paradox Valley from Coke Ovens, the first mine on the ridge that bounds the valley on the south is the Thunderbolt, from which considerable ore has been taken. The mine is several hundred feet above the bottom of the valley and can be reached only by trail from the main road. The ore must be packed on burros to the foot of the hill and there transferred to wagons. About 2 miles down the valley, on a continuation of the same ridge at a height of 700 feet, is the main camp of the General Vanadium

A. CLIFF MINE, SAUCER BASIN, PARADOX VALLEY. ANOTHER VIEW OF 4-FOOT STRATUM OF CARNOTITE AND VANADIUM ORE.



B. ELECTROSCOPE FOR DETERMINING THE RADIUM CONTENT OF AN ORE.

Co. Operations had ceased for the winter on December 1, 1912. From most of the deposits at this place all the ore that can be shipped at a profit under present conditions has been taken. There is, however, a large quantity of low-grade ore in place besides what is on the dumps. No water is available for concentrating, the supply for the camp having to be brought by team and burro at a cost of $1.25 per barrel.

The Jo Dandy, behind the camp and a little higher on the ridge, was until recently a heavy producer, but is now nearly exhausted. There are six or seven openings to the workings. The ore is in many ways similar to that of the McKeever prospect on the opposite side of the valley. A "bug hole" contained some blue-black and red vanadium ore. The surface of the sandstone on the sides of one of the openings is stained dark green, light green, purple, and orange yellow, the latter tint predominating, but the sandstone is almost white below. Gypsum is found throughout the mine, layers from an eighth to a quarter inch thick being common, and surrounds some of the "bug holes." There is considerable low-grade ore left in the mine, besides what is on the dump.

West of the Jo Dandy is the Blackbourne claim, where a number of drifts follow an ore body that outcrops. All shipping ore has been removed from these drifts, but bands of low-grade ore, about 2 feet thick, outcrop almost continuously for a distance of more than 200 yards.

The Valley View, Opera Box, and Kent Smith claims lie west of the Blackbourne in the order named. The Standard Chemical Co. owns one, the General Vanadium Co. owns the other two. The three claims are 200 feet above the Blackbourne, the duplication of the ore-bearing zone being due to a "slide," which is visible all the way to the Monogram claim, 2 miles farther west. In the Valley View claim the ore body dips 10°, then 30°, and then pinches out.

The ore left in these claims is low-grade vanadium ore that carries a little carnotite, usually as thin layers. All three claims have been good producers of vanadium and have also yielded a considerable quantity of carnotite.

Two miles west are the Monogram, Greenback and other Standard Chemical Co. claims, which are reached from the camp by a trail along the ridge. Also a steep trail, along which ore is packed, ascends from the valley below. The Monogram and Greenback No. 3 are both on a second "slide" and about 150 feet below the mines on the first "slide." Much of the overburden has been removed and practically all of the ore taken out. West of the Monogram and at the same level is the Quarter Circle, where a cut into the hillside yielded 100 sacks of good ore. Greenback No. 2, above Greenback No. 3, was nonproductive and Greenback No. 1, above Greenback

No. 2, yielded only 80 sacks of ore which was not high grade. West of the Greenback claims and above the Quarter Circle is the Bobtail, on which two open cuts produced some medium-grade ore. The ore body, however, pinched out. Farther west is the Last Chance, and west of this claim are the Happy Thought, Little Tom, and Annie May claims. Some good ore is exposed on the Annie May, although 150 sacks have been taken out. The overburden is heavier than on most of the claims. On practically all there seems to be a direct relation between the thickness of the overburden and the quantity and grade of the ore, the latter improving where the overburden is thicker. This relation is discussed in the chapter on the origin of the deposits. The ore body in the Annie May is in the form of a series of sharply-dipping steps. On the Holliday, below the Annie May, is a tunnel that connects with a 30-foot shaft from the Annie May. There was no ore in the tunnel, but some carnotite showed in a crosscut. Some medium-grade ore was taken from the Wilson, to the southwest. Still farther southwest are the Jasper and Greystone claims on which assessment work only has been done.

On all of these claims the ore is in pockets. Much of the shipping ore has already been removed.

The Standard Chemical Co. has operated many of its claims during the year, but practically all of the ore mined in 1912 was stored at its headquarters at Coke Ovens.

BULL CANYON, COLO.

Bull Canyon lies nearly due south of Paradox Valley and 8 miles from the Monogram mine. A rather rough trail ascends the ridge on the south side of Paradox Valley for 500 feet and then descends more than 1,000 feet into a long, narrow canyon which ultimately leads into Bull Canyon. The scenery is wild and the topography rugged. The canyons are bordered by high steep cliffs of red sandstone capped with the white sandstone in which are the uranium deposits. Some ore is taken out by way of Monogram camp on pack animals and transferred to wagons in Paradox Valley, and some is packed several miles over another trail and a road to Redvale, whence the ore is taken to Placerville.

The camp at the Cummings claims is well situated, and about 12 men have been employed. Operations began in October, 1912. The claims are rather scattered. The Black Fox is about 4 miles down the canyon from the camp. The outcrops are in the rim rock 150 feet above the canyon. There is a relatively thick overburden of sandstone containing four thin streaks of blue clay. On the south side of the ore body a sandstone roll cuts across at an angle of 45°. The ore extends to this roll, but not beyond it. At the top of the pocket a "bug hole" 3 inches in diameter carried vanadium ore, dark blue, with some red splotches. The main body of ore, 3 feet thick, was largely a mixture of blue and greenish-yellow vanadium and uranium ores carrying red streaks of calcium vanadate. The dip is about 10° N. About 8 feet north of the pocket the calcium vanadate disappears and the ore is yellowish green on top and black mingled with blue below. At this point the ore body was 1 foot thick, but the bottom had not been exposed. Forty sacks had already been taken from the prospect.

Two and one-half miles northeast of the Cummings camp, on the west side of the canyon and some 300 feet above the bottom of the valley, is the Boot Leg claim, on which some carnotite is exposed. Seventy feet higher is the Widow claim, on which prospect work has just begun. There are indications of carnotite in a greenish-yellow sandstone that outcrops on the hillside. Overlying this sandstone is another which at first sight looks much like decomposed granite. At the Fawn claim, across the canyon from the Widow, some carnotite is exposed, but only prospect work has been done.

On the Bob-o-link and Sundown claims, about 1 mile from the Cummings camp, is a fair showing of both carnotite and a dark vanadium ore. Some ore has been shipped from the Bob-o-link.

The Wedding Bell claim, near the Black Fox, has produced some good ore by careful sorting.

In Bull Canyon are also the General Vanadium Co., Saunders, and Cloud claims.

MCINTYRE DISTRICT.

The McIntyre district, which lies south of Bull Canyon and was the scene of some of the earliest mining for carnotite, was not visited by the writers. In this region the American Rare Metals Co., of Denver, has its plant. We are informed that the work done has made only small inroads on the ore supply. The larger part of the ore is rather low grade. Shipments have to be made by way of Dolores.

HYDRAULIC, COLO.

The General Vanadium Co. during the year opened a number of prospects at Hydraulic, and is now working at that place an aerial tramway across the river, saving a haul of several miles.

ORIGIN OF THE DEPOSITS.

It is difficult to form a definite opinion as to the origin of the carnotite and vanadium deposits. Hillebrand and Ransome[a] show that the ores must have been carried to their present position and that the vanadium and uranium compounds could not have been the original cementing material of the quartz grains, but in all probability locally replaced the calcite that forms the matrix of the ordinary light-colored sandstones in which the ores occur. They express the opinion that the carnotite resulted from local concentration of material already in the sandstone, and that its deposition as carnotite was under conditions determined by proximity to the surface and probably was partly dependent on a semiarid climate.

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[a] Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, p. 17.

Hillebrand[a] has shown that in small amounts vanadium is widely distributed in sandstones, limestones, and igneous rocks. Occasionally the proportion of vanadium in a sandstone is such that the material may be worked commercially at a profit. In the roscoelite-bearing sandstone deposits around Newmire, Colo., the ore mined contains on an average 1½ per cent V_2O_5, and in places as much as 2½ per cent V_2O_5. In the newly discovered silver and vanadium bearing sandstone in Eagle County, Colo., some of the ore contains 2½ per cent V_2O_5. It is therefore not difficult to state a possible origin of the vanadium that has been concentrated in the carnotite deposits. Almost invariably vanadium minerals are associated with the carnotite.

To explain the origin of the uranium is more difficult. No pitchblende, even in small quantities, has been found near these deposits, Gilpin County, several hundred miles away, being the nearest locality where pitchblende occurs. In the absence of definite knowledge as to any deposit from which uranium could have been derived, it seems reasonable to believe that the uranium came from sandstone overlying or underlying the ore bodies, having been leached from these sandstones and concentrated with the vanadium. In some cases it has been noticed that the ore is underlain by an impervious blue clay, which may have been a factor in determining the concentration of the uranium compounds. This is particularly true at Thompsons.

The so-called "bug holes" appear to have escaped the notice of Hillebrand and Ransome. Many of these holes are 30 to 40 feet long and 2 to 5 inches in diameter; the walls are usually incrusted with quartz or gypsum. Almost invariably these holes run downward at a slight angle into the upper parts of an ore body, although a few enter the lower part of a deposit, and end abruptly in the ore. They are filled with high-grade ore, usually carnotite, although in some the blue and black ores of vanadium predominate. The other end of these "bug holes" opens into a funnel-shaped mass of soft sandstone, heavily impregnated with ore, that grades into the country rock. The appearance of one of these "veins" is that of a funnel with a long stem. Undoubtedly these holes represent channels through which ore-bearing solutions were transported. How far the ore-bearing solution traveled and whence it came are questions more difficult to answer.

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[a] Hillebrand, W. F., Distribution and quantitative occurrence of vanadium and molybdenum in rocks of the United States: Am. Jour. Sci., ser. 4, vol. 6, 1898, pp. 209-216.

We collected a number of samples of the overlying sandstones, which appeared to be perfectly free from uranium stains of any kind. Determinations of the radioactivity of some of these samples are given below:

Radioactivity of samples of sandstone.

1. Sandstone 3 feet above ore, Wilson mine, Saucer Basin: 1 gram contains 3.7×10^{-12} gram of radium.
2. Sandstone 2 feet above ore, Long Park: 1 gram contains 261.5×10^{-12} gram of radium.
3. Sandstone 3 feet above ore, Skull Creek: 1 gram contains 262.8×10^{-12} gram of radium.
4. Sandstone 3 feet above ore, Black Fox claim, Bull Canyon: 1 gram contains 9.4×10^{-12} gram of radium.
5. Sandstone from sandstone roll, Black Fox claim, Bull Canyon: 1 gram contains 105.4×10^{-12} gram of radium.
6. Sandstone 3 feet above ore, Telluride No. 8, Thompsons: 1 gram contains 2.9×10^{-12} gram of radium.
7. Sandstone 1 foot above ore, Telluride No. 8, Thompsons: 1 gram contains 23.5×10^{-12} gram of radium.

There are very few sandstones in other localities having a radioactivity of more than 2×10^{-12} gram of radium. That of the sandstone of the Desert of Sahara is 0.4×10^{-12} gram of radium. A fair average of all sandstones is probably about 1×10^{-12} gram of radium. The results given above show that the radioactivity of the sandstones tested is 3 to 262 times the average. On account of this large variation it is difficult to draw definite conclusions, but the figures show that in many cases the sandstone 2 to 3 feet above the ore contains a quantity of radium, and therefore of uranium, many times the normal. Sample 5 was taken from a sandstone roll that cut off rich ore on the Black Fox claim. This sandstone was favorably situated to receive leachings from the rich ore, and yet it is less than half as active as some samples of sandstone collected 2 and 3 feet above ore. On the other hand, the fact that on the south side of East Paradox Valley the thickness and richness of the ore deposits seem to vary directly with the thickness of the overburden appears to lend some weight to the theory of downward enrichment. Further examination of the country rock will probably throw light on the origin of the ore deposits. The results already obtained, however, seem to indicate that the uranium was disseminated in the sandstone country rock and has been concentrated in ore bodies by the action of water, the "bug holes," in some cases at least, acting as channels for the ore-bearing solutions.

MINING METHODS AND COST OF MINING.

Simple methods have been employed in mining carnotite, and so far most of the surface deposits have been worked by open cuts. At a few places drifts and adits have been run to follow ore into a hillside. Prospecting with drills has not been done, so far as we could ascertain, but drills will have to be used later when the surface deposits have been exhausted. At the Cliff mine in Saucer Basin a number of drifts and crosscuts have been driven into the cliff side; in most of them there is some timbering, but as a rule the roof is firm enough to stand without support. At another claim a deep shaft is sunk to a drift, both shaft and drift being well timbered.

Only hand drills are used in mining. The usual explosive is a "40 per cent" dynamite, which does not shatter the rock too much. Excessive shattering of the rock would cause the loss of much valuable mineral as fine dust. To reduce losses in the dust and fine ore, care must be taken in sorting.

The exact cost of mining is not easy to ascertain, as detailed accounts are not kept by the average miner, but a safe estimate, based on information received from various persons, is that the cost including hand sorting, exceeds $20 and perhaps averages $30 per ton of shipping ore. In one case the mining, sorting, and sacking of 1 ton of carefully mined shipping ore required the services of 5 men for 3 days, the men being paid $3 each per day. Therefore, the labor cost of this ton of shipping ore was $45. To this cost must be added the cost of sacks, which is $3 per ton of ore when single bags are used. The mining and sorting cost for this ton of ore seems to be unusually high, and a cost of $30 per ton would be nearer the average. The cost of mining will increase considerably with the depth of the workings, especially as it is improbable that any extensive single deposit will be discovered, and the prospecting and development of small deposits are expensive.

As a rule the mine operators keep no detailed account of mining and other costs. Such an account should be kept, in order that the actual profit per ton of ore mined may be arrived at and the efficiency of operations estimated. As a suggestion in the interest of greater efficiency, a short outline of what properly kept accounts should show is given herewith:

The actual cost of mining and sorting per ton of ore.

Cost of powder, fuse, and tools per ton of ore.

Cost of bags, sacking ore, and sewing bags.

Cost of hauling.

Cost of freight.

Pro rata cost of management, charge for depreciation of investment (property and equipment), assayers' charges, ore sampling charges, etc.

In case the ore is milled, the accounts should show the cost of concentration per ton of ore, including the cost of fuel, water, and haulage, as well as charges for amortization of plant, repairs, etc.

The sums expended in establishing a camp and in prospecting for ore should be considered and charged against the tonnage obtained from properties found to be valuable. As an example, let it be assumed that an ore contains 2.5 per cent U_3O_8 and 4.5 per cent V_2O_5, with a value of $97 per ton, and that the costs for each ton of marketable ore are as follows: Mining and sorting, $30; powder, tools, etc., $2; bags, sacking, etc., $4; hauling, $20; freight, $14.50; then the actual cost per ton is $70.50. If 200 tons of marketable ore are mined yearly $10 per ton should be added for management and other charges, making the total charge per ton $80.50. The net profit is $16.50 per ton of ore shipped.

If low-grade ores are concentrated, the calculation of cost must include charges for concentration, amortization, etc., at the rate of about one-half ton of shipping ore recovered for each ton of shipping ore hand sorted without concentration, plus the extra expense, if there is any, for mining low-grade ore.

TRANSPORTATION AND PRICES.

The cost of getting the ore to market from the different localities varies considerably. Unfortunately practically all of the ore deposits are miles from a railroad. Some of them are so far away that it is impossible to sell the ore at a profit at the present time.

At Meeker mining operations have not yet reached the stage where ore can be sold at a profit. It costs between $20 and $25 per ton to deliver the ore at Rifle. To this must be added freight charges, which are about $11 to New York.

The cost of transportation from the Skull Creek deposits is prohibitive, except for high-grade ore. It costs $20 per ton to carry the ore to Vernal, $15 per ton from Vernal to Watson, and $8 per ton from Watson to Mack. The freight rate from Mack to New York is about $12 per ton.

The cost of transportation for the Green River ores is probably less than that for any other of the carnotite deposits. The cost of the haul from the Lorimer camp to the railroad is $4.50 per ton and the freight rate is $13 to New York.

The rates at Thompsons are a little higher. Not enough work has been done to be able to give an exact figure, but the haul from the mines to the station would probably be about $6 per ton and the freight rate practically the same as that from Green River.

The cost of haulage varies widely in Paradox Valley. This difference depends more upon whether pack trains have to be used in addition to wagons than upon the actual distance of the mines from Placerville. At Long Park very little ore has to be packed and most of it can be loaded directly on the wagons, the cost of hauling to Placerville being \$20 per ton. Saucer Basin has to stand this charge for hauling and in addition a pack rate of \$8 to \$10 per ton. It is somewhat difficult to find out the exact cost of packing the ore on burros, as many of the companies own their own animals, but the figures here given may be considered approximately correct. From the East Paradox mines of the Standard Chemical Co. and the General Vanadium Co. the cost of hauling is \$18 per ton; in addition the ore has to be packed down the mountain side at a cost of probably \$2 or \$3 per ton. From Bull Canyon the cost of hauling is at present \$20 per ton, with a pack rate of \$5 per ton. This figure will probably be reduced slightly after the completion of a road that is being built. The freight rate from Placerville to New York is \$11.57 per ton, and from Placerville to Hamburg or Liverpool, via Galveston, \$14.50 per ton. The average cost of transportation of the ore is, therefore, \$29.57 to \$42.50 per ton. In the case of some of the outlying districts the cost per ton is even more than the higher figure. Adding to this an average cost of \$34 per ton for mining, sorting, and sacking, the total cost to the operator is from \$63.57 to \$76.50 per ton. This does not include costs of powder, tools, assayer's charges, etc.

Up to the present time there has been a ready market for ore containing 2 per cent U_3O_8 or more. Most of the purchasing agents have refused to take any ore of a lower uranium content, although a small quantity of ore has been sold during the past year containing 1.7 or 1.8 per cent. All ore is bought on the basis of its uranium content, a high-grade vanadium ore being difficult to sell if it contains less than 2 per cent U_3O_8. This is one of the unfortunate features of the present mining conditions, much vanadium ore carrying a little uranium being left on the dump and in the mine. One carload of ore that contained 8 per cent V_2O_5 but only 1½ per cent U_3O_8 was held up indefinitely. If this ore had carried 2 per cent U_3O_8 and less than 3 per cent V_2O_5, it could have been readily sold. The ore is purchased for the radium that it contains and not for the uranium and vanadium, which are considered as by-products. The foreign buyers established the 2 per cent minimum for uranium and seem to be indifferent to the percentage of vanadium in an ore.

The prices paid for ore vary within narrow limits. One agent offers for 2 per cent U_3O_8 ore \$1.30 per pound of uranium oxide; for 2½ per cent ore, \$1.40; and for 3 per cent ore, \$1.50. For the V_2O_5 content he pays \$0.30 per pound. These prices are f. o. b. New York. An operator who has received offers from several agents states that the prices quoted vary from \$1.25 to \$1.40 per pound of uranium oxide for 2 per cent ore and \$0.35 per pound of vanadium oxide for ore containing more than 3 per cent of this oxide. Some operators sell their output entirely upon the basis of its uranium content and get nothing for the vanadium. Such a basis usually prevails where the ore is fairly high in uranium and low in vanadium. Where the ore is high in vanadium it is sold for both its vanadium and uranium content according to the prices stated above. When the vanadium is not paid for, the average price given for ore containing 2 per cent U_3O_8 is $2 per pound and for 3 per cent ore $2.25 per pound. Little more is offered for ore containing 3 to 5 per cent U_3O_8, but for ore containing more than 5 per cent the rate is higher. The high-grade material from "bug holes," carrying 12 to 20 per cent U_3O_8 or more, brings about $3 per pound of the oxide. These prices are all f. o. b. New York or Hamburg.

Deducting the costs of mining and transportation from these prices leaves a very small profit for 2 per cent U_3O_8 ore. One unfortunate feature of the system of marketing is the fact that much ore passes through the hands of four or five agents before it reaches the final purchaser. With better mining conditions and a lower cost of production, the elimination of some of the middlemen, and with prices for ore based on the fact that radium is the main valuable constituent, the operator should be able not only to get a reasonable profit on 2 per cent ore, but also to realize a small profit on ore containing 1½ per cent U_3O_8, especially if it is high in vanadium.

CONCENTRATION OF ORES.

NECESSITY FOR CONCENTRATION.

In the course of investigations of these uranium and vanadium ore deposits it was found that much low-grade ore was left in the mines, and thus lost, and that there was great loss in hand sorting. Since the market demand is for a material containing at least 2 per cent uranium oxide, the miner, that he may ship no ore below this limit, eliminates all ore that in his opinion contains less. As a result much low-grade and also some shipping ore are left in the mine or are thrown on the dump. By concentrating the low-grade ores at or near the mine these wastes can be reduced. There is no doubt that the ores can be concentrated. Tests made on a small scale substantiate this statement.

In concentration by mechanical means the dry as well as the wet method may perhaps be used to advantage, as little or no water is available in many places where these ores occur. As formerly stated, most of the deposits are far from a railroad, so that wagon hauls are long and expensive. By concentration the bulk of low-grade ore to be hauled can be reduced and at the same time the percentage of mineral in the concentrate raised to a marketable point. Thus a saving in hauling and freight rates is effected, and the concentrate on account of being richer should bring a better price. Low-grade ores thrown on the dump or left in the mine constitute a natural waste. Much of the low-grade material left in the ground during former operations and perhaps much of the dump material can never be recovered. Much of the ore thrown on the dump disintegrates on exposure to the air, so that much of the valuable contents is washed or blown away. Some of the minerals in the dump rock are liable to be leached out by the rain.

WET CONCENTRATION.

The carnotite ores are of great importance, especially on account of their uranium and radium content. Carnotite, which contains both uranium and vanadium, is a yellow, crystalline, pulverulent material, with a specific gravity of 4.136. It occurs in white sandstone as an incrustation on the faces of joints and fractures and is deposited around and between the individual grains of the sandstone, often strongly adhering to the even surface of a grain. The size of these grains varies greatly, but is generally from a little less than 0.1 to 0.2 mm. in diameter.

In concentrating such an ore the rock should be broken and the pieces reduced to about 40 to 80 mesh with crushers or rolls, the crushing being accompanied by rubbing in order to loosen the carnotite from the sand grains. Further attrition should be applied to the 40-mesh material by convenient means. Fine grinding should be avoided, as it produces too much slime.

If the crushed and rubbed material is washed in a revolving tank or trommel provided with a rubbing device, the particles of carnotite that are in suspension can be drawn off after the grains of silica have settled in the tank or cylinder. As some of the carnotite is carried down by mechanical action with the silica in settling, more water should be added and the operation repeated as many times as may be profitable. With such a treatment a large part of the valuable minerals can be extracted.

The slime is washed into settling tanks or other suitable device, and the water after the settling of the slimes can be used over again, as will be necessary in the arid districts, where most of the deposits are located and where water is scarce. Enough water can be collected and stored away for this purpose from seepage, springs, and during rainy days, and in many cases a wet treatment can in this way be made possible.

RESULTS OF WET CONCENTRATION TESTS.

Tests have been made of ore taken from the dumps at various claims. In one instance this ore contained 2.02 per cent U_3O_8 and 2.32 per cent V_2O_5. Concentration by the wet method gave the following results:

Results of concentration by wet method.

	Quantity.	V_2O_5.	Weight of V_2O_5.	U_3O_8.	Weight of U_3O_8.
	Grams.	*Per cent.*	*Grams.*	*Per cent.*	*Grams.*
Ore	1,943	2.32	44.08	2.02	39.348
Concentrate	191.3	9.92	18.976	8.84	16.91
Residue	1,751				
	1,942.3				

Loss in concentration, 0.7 gram. Concentration ratio=10.156:1. Extraction from ore by concentration, V_2O_5=43.035 per cent; U_3O_8=43.08 per cent. Ratio of ore to concentrate, vanadium content=1:4.275; uranium content=1:4.376. Ratio of radioactivity, measured in the electroscope, ore to concentrate=1:4.56, which shows that the concentrate is 4.56 times more radioactive than the ore.

Other tests made with ore containing less U_3O_8 and V_2O_5 have shown that an extraction can be made with a proportionately similar result.

The following result was obtained with a reddish-brown vanadium-bearing sandstone from Utah. The ore was crushed to the size of the grains, rubbed, and then washed in cold water. The slimes thus obtained were settled and decanted or filtered.

Result of concentration of vanadium-bearing sandstone by wet method.

	Quantity.	V_2O_5.	Content of V_2O_5.
	Grams.	*Per cent.*	*Grams.*
Ore	400	5.57	22.28
Concentrate	85	17.35	14.75
Residue	315		

Concentration ratio=4.72:1. Extraction=66.2 per cent.

According to electroscopic measurements, and by comparison with a standard material of known U_3O_8 content, it was found that this ore contained approximately 0.71 per cent U_3O_8 and the concentrate contained 1.62 per cent.

In concentrating this vanadic sandstone the content of vanadium was therefore raised to 17.35 per cent V_2O_5 and the concentrate can be sold in competition with the Peruvian patronite. In addition the uranium and the radium can be easily extracted from this ore. Some of the vanadium can not be extracted mechanically; it probably exists in the sandstone as roscoelite, which is bound in the silica.

DRY CONCENTRATION.

Where water is scarce, a dry process can be adopted for low-grade ores. As good concentration with a dry as with a wet process can not be expected, but an equally efficient if not a better extraction can be obtained.

In a dry process the rock should be reduced with crushers and rolls as nearly as possible to the size of the grains of the sandstone. Much of the carnotite can be obtained by sifting the crushed material through a 120 to 150 mesh screen. Care must be taken to collect all of the dust, as this is richest.

Good results can be obtained by using an air current to blow the finest particles into a dust chamber, in which they are collected. The air current must be steady and evenly expanded, so that the coarse particles will drop out of it, and strong enough to completely lift the material. The coarse material should be carried along with the current for some distance, so that as much as possible of the powdery material can be freed and blown from the coarse grains, which should finally drop without carrying the powder with them. These particles should fall into a discharge spout and thence into bins or other suitable device. The powdery material, which carries the valuable minerals, is collected in dust chambers. These chambers must be perfectly tight. Much valuable mineral has been lost in sampling shipments when the sampling device had no arrangement for saving the dust.

A number of machines that doubtless could be used for the dry separation of these ores are now on the market.

Another method of dry concentration of low-grade ores consists of simple sifting, with previous rubbing, of the crushed ore. The ore is reduced with crushers and rolls to about 40 mesh and is then passed over an oscillating 150-mesh screen. The undersize from the 150-mesh screen is concentrate. The oversize is carried to a device in which the sand grains are subjected to a thorough rubbing, which removes from the silica the adhering fine particles of carnotite.

In the experiment stiff wire brushes rubbing against a steel plate were used with success for the attrition. The entire material was then brought over two screens, one overlying the other; the upper screen being 80 mesh and the lower one 150 mesh. The table below clearly shows what results can be obtained by such a process. The screens must be encased and all of the dust must be collected. The dust from the sifting operation and that which remains in suspension in the dust-proof screen boxes can be blown from these boxes by suitable means into dust chambers and then added to the concentrate.

A concentration by this dry method on a small scale gave the following results:

Results of dry concentration by sifting with previous rubbing.

	Mesh of sieve.	Quantity.	V_2O_5 content.	Weight of V_2O_5.	U_3O_8 content.	Weight of U_3O_8.
		Grams.	*Per cent.*	*Grams.*	*Per cent.*	*Grams.*
Ore	40	500	2.40	12.25	2.24	11.20
Concentrate	Under 150	134	4.66	6.24	4.42	5.92
Residue	Over 150	169	1.60	2.70	1.53	2.58
Do	Over 80	197	1.50	2.95	1.37	2.68
		500		11.89		11.18

Concentration ratio—ore: concentrate—3.73 : 1; extraction V_2O_5=50.85 per cent; extraction U_3O_8=52.85 per cent.

Care must be taken in crushing the ore for either wet or dry concentration that little, if any, of the dustlike material is lost. Dust chambers should be attached to the crushers, or the latter should have air-tight casings and the material should not be removed until the dust has settled.

COST OF CONCENTRATION.

The factors to be considered in a calculation of the cost of concentrating the ore by a wet method are:

1. Cost of concentration:
    - (*a*) Water supply.
    - (*b*) Cutting wood for fuel for power and drying (or cost of gasoline if that is used for fuel).
    - (*c*) Hauling of fuel or gasoline.
    - (*d*) Wages for concentrator men.
2. Amortization of equipment for concentration (pro rata of tonnage of material treated per annum for such equipment and for management).

Two men, at $3 a day each, should be able to operate a small plant. The cost of water supply should not exceed $2 a day. The cutting of fuel or cost of gasoline should not be more than $2 to $2.50 a day. The hauling of the fuel would probably cost $1.50 a day.

On an average, from 10 tons of ore mined only 1 ton of ore with a content of over 2 per cent U_3O_8 is obtained by hand sorting. From the other 9 tons, thrown on the dump, about 5 tons of low-grade ore can be separated for concentration. Assume that such low-grade ore contains, on an average, about 1 per cent U_3O_8 and 1.5 per cent V_2O_5. With a concentration ratio of 10 to 1, half a ton of shipping ore should be obtained. The total extraction of mineral by a wet method should be about 50 per cent, and the grade of the material obtained would therefore be fairly high. In an actual test on such a low-grade ore the percentage of U_3O_8 in the concentrate was found to be about 4.5. This should of course command a proportionately higher price.

The cost of concentration by the dry process is approximately the same, less the cost of the water supply.

The cost of concentration, including ample charges for amortization and for management, should not exceed $20 per ton of concentrate. These figures are conservative and probably the ores can be concentrated more cheaply.

The average cost of mining with present methods at the rate of 10 tons of ore a day with 6 men and the hand sorting done by the foreman is about $30 per ton of shipping ore. This figure does not take into account the powder used in blasting, the wear of tools, etc. If the low-grade ore is utilized by means of concentration, the expense of mining is thereby increased little, if any, and only the additional cost for the concentration has to be taken into consideration, the charges for sacking and hauling the concentrate to the station being the same as for a ton of ordinary ore.

From the above it will be seen that through utilizing the low-grade ores the average tonnage per annum could be increased 50 per cent with little extra expense. But as the uranium and vanadium content of the concentrate would be at least double that of the average ores shipped, the total production of uranium and vanadium from these ores would be doubled. For various reasons it is hardly to be expected that all of the waste material can be utilized, and therefore such a large increase will probably never be actually obtained.

Perhaps several of the operators could combine and install an equipment for mechanical concentration and treat their low-grade ores in a central plant, especially where several operators are in the same district and the claims are near each other. With such a plant in use the production of marketable ore at the present rate of mining could be increased by about one-half without increasing the cost of mining and with only the additional expense of concentration. This increase can be obtained by treating material that hitherto has been lost.

The assumption, which has been made by many of the dealers, that by concentration a large part of the radium content of the ore would be lost, is entirely without foundation, as no such losses of radium in the concentrate can occur from any mechanical treatment of the ore. This has been proved by our tests of the concentrates (pp. 37, 39). Undoubtedly the statement has reference to chemical concentration, in which, of course, such losses might occur.

POSSIBILITY OF CHEMICAL TREATMENT.

A chemical treatment of the low-grade carnotite ores at or near the mine would undoubtedly be better and a larger extraction could be obtained were it not that the present high cost of hauling chemicals to the mine prohibits such treatment. In addition, the average operator could not obtain such labor as would be necessary in a chemical process. The deposits worked by the smaller miners are scattered, and their probable output is too uncertain to justify the erection of even a small chemical plant. This statement does not apply to a chemical plant conveniently situated for the treatment of custom ores.

GENERAL STATEMENT.

The possibility of a continuous supply of carnotite ores in the future is naturally of great interest. In the Utah fields, as already stated, the ores are generally low in uranium, although occasional small pockets of high-grade ore are found. Some of these deposits have the advantage of short hauls; others, such as those at Table Mountain, have as long a haul as most of the ores taken out of Paradox Valley. Since the present demand requires an ore containing at least 2 per cent U_3O_8 the operators in the Utah fields have to do such careful sorting that a large proportion of the uranium ore is wasted. There is therefore, as already explained, great need of a concentration method, or methods, that can be used by an operator at his own camp, or at least by a group of operators working nearby claims. In this way not only would the low-grade uranium ores be made available but also the vanadium ores that contain too little vanadium to be worth marketing under present conditions. It is quite possible that such ores may be so concentrated that the concentrates can be handled for their vanadium content alone, even though they contain too little uranium to warrant their being classified as uranium ores.

In Paradox Valley at present little difficulty is experienced in obtaining 2 per cent ore by careful sorting, and a fair amount of ore carrying 3 per cent or even more has been mined during the past year; but there has been much waste in the sorting of this ore and more low-grade material has been left in the mines untouched. This is particularly true in Saucer Basin and at certain points on the south side of East Paradox Valley. It is difficult to state just how much ore suitable for concentration but not rich enough to ship on the present basis is available in the Paradox and surrounding districts. It is certain, however, that there is enough already in sight to double the output for several years to come, if it could all be made marketable.

As this is hardly possible a 50 per cent increase is probably all that could be expected under the most favorable conditions.

The present policy, which is really necessary on account of the cost of production, of shipping only ore containing 2 or 3 per cent U_3O_8, is bound to have a far-reaching result. The buyers are urging the shipment of even higher-grade ore and are offering a bonus for it. Hence the tendency has been to raise rather than to lower the minimum percentage, a correspondingly larger waste resulting. Although some of the pockets in Paradox Valley are large, a number of the largest have been worked out within the past two years. The output of 2 per cent ore may be as large for several years to come as it has been during the past year, and may even increase for a while, but after a few years the production of high-grade material will inevitably decline. The buyers will then either have to accept a smaller output or handle low-grade material. New pockets will naturally be opened from time to time and a certain percentage of high-grade material will be available for many years to come, but it is extremely important that methods be devised for utilizing at least some of the low-grade material now being thrown on the dump or left in the mine.

The United States possesses unique deposits in these carnotite ores. They constitute at present the largest known supply of radium-bearing minerals in the world. With the exception of the ore mined and utilized by two firms, practically every pound is shipped abroad. Up to the present very little interest has been shown by Americans in these deposits, which may not be duplicated in so far as quantity goes in any part of the world.

The only other large deposits of uranium-bearing ores known are those in Austria. They are considered of such importance that the Austrian Government has taken entire charge of them. The output from the carnotite fields of this country is much larger than that from the Austrian mines and is likely to continue larger for some time to come, but the ore should be mined with minimum waste and the industry should yield a maximum profit to this country.

There is little danger that the amount of ore mined per annum will increase materially. The long hauls and the scattered occurrence of the deposits make it difficult to largely increase the production in any one year. Such an increase can be accomplished only by utilization of the low-grade ores.

A word of warning may be added at this point. The fact that carnotite and its associated ores are found on a claim does not necessarily mean that such a claim is worth even the assessment work that has been expended on it. Already one company is making extravagant statements in its advertisements, and others may follow it.

PITCHBLENDE DEPOSITS.

DESCRIPTION OF DEPOSITS IN THE UNITED STATES.

Pitchblende has been found in the following localities in the United States: Feldspar quarry, at Middletown, Conn., in large octahedrons; in Hall's quarry, at Glastonbury, Branchville, Conn., in a pegmatite vein and usually embedded in albite; at Marietta, S. C.; in the Baringer Hill district, Llano County, Tex.; in the Bald Mountain district, Black Hills, S. Dak.; in Gilpin County, Colo.; and in Mitchell County, N. C.

NORTH CAROLINA DEPOSITS.

In Mitchell County, N. C., small quantities occur near Penland in the quartz and feldspar mines. The mineral is usually associated with quartz, but in places with orthoclase feldspar, and still more infrequently with albite. Its presence is usually indicated by a dull-green stain on the quartz or spar, although such stains do not invariably mean that ore is present. The pitchblende from near Penland is usually high grade and is often associated with yellow gummite. In probably the best mine only 50 pounds have been found in 1½ years, so that as yet the mining of these deposits can not be considered a commercial enterprise. The ore is usually sold in small quantities as museum specimens at specimen prices.

COLORADO DEPOSITS.

In Gilpin County, near Central City, Colo., are five mines that have produced pitchblende, namely, the Kirk, the Wood, the German, the Belcher, and the Calhoun. These mines are all within about 2 miles of Central City and are situated on or very close to Quartz Hill, at an altitude of 9,500 feet. The Kirk, Belcher, and German mines are close together on the hill; the Wood and Calhoun are in the valley.

These mines were originally gold mines and until recently have been worked mainly for gold. Gilpin County is the oldest mining district in the State of Colorado, and it was here that the first stamp mills and smelters in Colorado were built. The ores near the surface are mainly free-milling; lower down they change to sulphides. The veins are persistent, but the quality of ore often varies considerably within a short distance.

KIRK MINE.

The original workings of the Kirk mine are about 100 feet deep. The majority of these have been filled up. In this manner the larger part of the 500 feet of the lode to the east of the shaft has been worked out to the above depth, but the 1,000 feet to the west has not been opened except by prospect holes. An old shaft and part of the workings are still open. In this shaft and workings a considerable amount of pitchblende was mined in the early days. The miners did not know at the time what the ore was and it was wasted in an attempted treatment for gold. When the pitchblende was found the gold gave out, and therefore the owner left his shaft and started a new one 185 feet to the west, which is now the main shaft. The next owners knew the value of pitchblende and about 1898 took out several tons of good ore. When further development was unsuccessful, the mine was bought by the present owner. The mine i more than 400 feet deep. The vein strikes almost due east an west and dips about 80° S. The shaft follows the vein. The mai levels are at 97, 140, 200, 300, and 400 feet.

The Kirk lode is 3 to 6 feet wide, and seems to be a fissure vein i gneiss and mica schist. It carries gold, silver, and copper, in additio to pitchblende. The best pitchblende was found between the 140-foot level and the 250-foot level on the east side of the shaft. Or was also found at other places, notably just above the 400-foo level and in this level. The ore shoot seems to pitch from east t west. Except in one place, the ore was everywhere against th country rock on the hanging wall. There were practically n stringers, spurs, or other indications that the vein was near excep that the country rock carried some uranium. Much of the ore wa exceedingly rich, some assaying 60 to 80 per cent U_3O_8. At place this rich ore was a foot thick; one single piece was removed tha measured 2 feet 8 inches by 1 foot 4 inches by 1 foot.

Reliable data on the production of pitchblende from the Kirk min before the present ownership could not be obtained. Since th present owner has had the mine, in round numbers about 20 tons o ore with an average content of 35 per cent U_3O_8, and something ove 100 tons of ore with a content of 3 to 4 per cent U_3O_8 have bee mined. Most of this ore was produced in 1905–6, and practicall all of the high-grade ore was sold abroad.

Therefore at a time when the Austrian mines were apparentl the only producers of uranium ore the Kirk mine was sending pitchblende abroad and supplying a large proportion of the radium tha was sold on the open market.

During the last few years the Kirk has not been worked regularly; therefore the production of ore has been small.

Near the Kirk are the Jeannette and Hilda claims. Some assessment work has been done on them.

GERMAN AND BELCHER MINES.

The German and the Belcher mines are a short distance north of the Kirk mine. It has been difficult to obtain satisfactory and reliable information concerning the production of uranium from these mines previous to 1911. About three years ago the Belcher mine produced 1,600 pounds of ore carrying 30 per cent U_3O_8, and shortly afterwards about the same quantity was mined in the German. During this period both mines were run in a rather haphazard manner with inefficient equipment. Last year both mines were taken over by the German and Belcher Mines Co.

The shaft on the German mine is down 600 feet, but is blocked at the 400-foot level. The main levels are at 130, 250, and 400 feet. The vein dips south, but is nearly vertical, like that of the Kirk mine. The Belcher mine is in the same vein as the German mine, several hundred feet to the east. The shaft is down 200 feet, with levels at 120 and 180 feet.

The country rock in these mines is gneiss and mica schist, similar to that in the Kirk. Intrusive andesite and granite are common. Unlike the Kirk, thin stringers of pitchblende are sometimes found that lead to pockets of rich ore. In addition to the pitchblende, the ore contains iron and copper pyrites and lead and zinc sulphides carrying gold and silver. The proportion of pyrites is larger than in the Kirk mine, and the ores carrying small quantities of pitchblende are much richer in sulphides than similar ores from the Kirk. Rickard[a] states that at places a "gouge" or clayey salvage mixed with soft pitchblende ore of low grade is found. This is supposed to be the result of crushing of the vein material during a second period of rock movement. As in the Kirk, pitchblende and gold do not occur together.

There is a good showing of pitchblende in the western part of the 130-foot level. The high-grade ore is from 1½ to 2 inches thick, some of it containing over 80 per cent U_3O_8. Lower-grade material is also found in the eastern part of the 250-foot level, indicating that the ore shoot pitches east. This is exactly opposite to the apparent pitch of the ore in the Kirk mine.

The company is doing development work chiefly at present and is working more than 20 men in two shifts, is using air drills, and has contracted for a considerable amount of diamond drilling.

The total production of pitchblende ore from the two mines from the fall of 1911 to January 1, 1913, has been 240 pounds of high-grade ore containing more than 70 per cent U_3O_8, 220 pounds of ore containing 20 per cent U_3O_8, 5 tons of ore carrying 2.6 per cent U_3O_8, and 1 ton of ore carrying 2 per cent U_3O_8.

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[a] Rickard, Forbes, Pitchblende from Quartz Hill, Colo.: Min. Sci. Press, June 7, 1913, p. 851.

THE CALHOUN MINE.

The Calhoun mine, south of the Kirk and German mines, is at present worked for gold only. There are two shafts on this claim, but the east shaft is not used at present. The mine has been a small producer of uranium. During 1912, $1,300 worth of pitchblende was sold for specimens—the total output during the year. This was nearly all high-grade material and was obtained at a depth of 387 feet. That portion of the mine from which the pitchblende was obtained is closed until a successful process for the concentration of low-grade ores has been developed.

THE WOOD MINE.

The Wood mine at present is being worked under lease by three men, who have had charge about a year. As in the case of the Calhoun, the gold is sought rather than the pitchblende. During March, 1913, a small pocket of pitchblende was struck, from which 400 pounds of high-grade ore were taken. The vein is narrow, being only 9 to 18 inches wide, but at places the ore is rich in gold. In addition, the ore carries lead, copper, silver, and near the west end of the 200-foot level, also zinc. There are two shafts on the property, with levels at 135, 160, and 200 feet. Most of the ore above the 135-foot level has been stoped out. During the past 5 years about 1 ton of medium-grade pitchblende ore has been mined. This was found a little below the 160-foot level, close to where ore has recently been found.

In general these mines are similar as regards the character of the veins, the quality of the pitchblende, and the depth at which the latter is found. A porphyry dike outcrops at several points just west of the pitchblende mines, and the workings of the Calhoun mine extend from the west shaft into this intrusion. The Alps mine, which is close to the Kirk and German, but which has never produced any pitchblende, lies west of this dike, which may have had something to do with localizing the deposition of the pitchblende. The sulphides, which are found in large quantities in these deposits, are doubtless due to secondary mineralization.

From the past showing of these mines, especially of the Kirk mine, one would be justified in classifying the veins at Quartz Hill as being among the important pitchblende deposits of the world.

WASTE OF LOW-GRADE ORE.

Until recently low-grade material has been largely neglected. Nevertheless the future of these mines depends largely on the utilization of this low-grade material. It is quite probable that much of the rejected material is low-grade uranium ore. Some rejected pieces picked up in the levels of the German mine carried 12 per cent U_3O_8.

If such material as this was discarded, it is more than probable that a large proportion of lower grade ore was thrown out. A composite sample, taken by one of the writers from eight different points in the dump of the old workings at the Kirk mine, assayed 0.8 per cent U_3O_8. This sample, of course, was not by any means an average of the whole dump, but it shows that low-grade material has been discarded in the past.

CONCENTRATION OF LOW-GRADE PITCHBLENDE ORES.

Attempts at concentrating the low-grade pitchblende ores have been made by various parties with fair results.

The ore should be hand sorted before treatment by mechanical means, to remove as much of the waste rock as possible. After crushing to the desired mesh, the material should be carefully sized and classified. This is essential, as a high concentration with a small loss of valuable minerals can not be obtained with unsized material. The order of the important compounds in the ore, in regard to their specific gravity is uraninite, pyrite, copper pyrites, and quartz. The difference in specific gravity of pyrite and copper pyrite is small, between pyrite and uraninite considerable, and quartz, being the lightest material, can easily be separated. In one concentrating test both a jig and an oscillating table were employed, and, seemingly, a better separation was obtained with the jig than with the oscillating table. The separation of the uraninite from the pyrite and copper pyrites can be accomplished by flotation, but better results can perhaps be obtained by magnetic separation after a slight roasting of the ore.

The pyrite and copper pyrites must be saved, as they may contain gold.

There is no doubt that a high concentration can be accomplished with little loss. Results obtained by the Bureau of Mines, which is working on the problem of concentrating low-grade uranium ores, will be published in another report. It is expected that ore containing as low as 0.5 per cent U_3O_8 can be utilized. This should help to make mining for pitchblende a better commercial venture, especially as there is a good yield of gold and silver to be derived from by-products.

Mechanical concentration of low-grade pitchblende ores has been done in a similar manner in Austria and Germany for several years, but no definite data in regard to methods and results have been obtained.

EUROPEAN PITCHBLENDE DEPOSITS.

A comparison of the Colorado pitchblende deposits with those of Europe may be of interest, although up to the present the former have not received much attention. The important European deposits are found in Germany and Austria. The ore deposits at St. Joachimstahl, Austria,[a] are in mica schist interbedded with lime schist and crystalline limestone. Toward the east and northeast the formation is gneiss. The gneiss was intruded by quartz porphyry subsequent to the deposition of the vein material. In the mica schist are fissures filled with volcanic material which cut the mineralized zone at various points and depths. The veins are usually 6 inches to 2 feet wide, in rare cases widening out to 3 feet. The mode of mineralization varies greatly. The ores are in both stringers and pockets, and contain the following metals: Silver (metallic and as argentite, polybasite, stephanite, tetrahedrite, proustite, pyrargyrite, sternbergite, and other minerals); nickel (nickelin, chloanthite, and millerite); cobalt (as smaltite, bismutosmaltite, and "absolan"); bismuth (as metallic bismuth, bismite, etc.); arsenic (as metallic arsenic, arsenopyrite); and uranium (as uraninite, or pitchblende and other alteration products). Galenite, zincblende, pyrite, marcasite, and copper occur in minor quantities.

The ore beds show that deposition took place in three periods. The cobalt and nickel were deposited first, then the uranium was deposited, and lastly the silver. In some cases the uranium ore is partly replaced by a dark-violet bituminous fluorspar.

Pitchblende is found at Joachimstahl, in Austria, and at Johanngeorgenstadt, Marienberg, Freiberg, and Schneeberg, in Saxony, and Přibram, in Bohemia, having a similar origin. Dolomite spar is always present, which has generally a white or yellowish-white color, but changes to a peculiar brownish-red hue where pitchblende begins to appear, and is a dirty gray where it is actually in contact with the ore. Deep-blue fluorspar is always present. Copper pyrite is found in small crystals and masses throughout the pitchblende. Pitchblende is also frequently found disseminated in small grains through a part of the mica schist forming a low-grade ore.

The mines at Joachimstahl have been worked since 1517. In 1545 the production of silver ores declined considerably, but since then the deposits have been mined for bismuth and cobalt. During the last 10 years the mines have been worked for uranium. The Edelleut Stollen has been exploited exclusively for uranium ores and the Austrian Government has erected a factory at Joachimstahl for the handling of these ores.

In the vicinity of Annaberg, on the Saxony side of the Erzgebirge, the silver-cobalt veins resemble those at Joachimstahl. A large number of these veins are in the mountains close to the town. Tin, lead, and copper pyrite are among the minerals which have been mined in the older formations. The more recent deposits contain silver and cobalt, or iron and manganese. The gangue is composed of barite, fluorspar, quartz, dolomite, and pyrites. There is also some amethyst, calcite, aragonite, kaolinite, and gypsum. The ores are cobalt, silver, and bismuth, with some copper pyrites, galenite, pitchblende, gummite, uranochalcite, and arsenic. There are similar deposits at Johanngeorgenstadt, Saxony. The mountain sides are filled with a network of veins. These veins contain tin and silver-cobalt ores and their strike varies greatly. Where dolomite spar is found, the silver-cobalt ores contain pitchblende, as at Annaberg. In the Gottessegen mine the pitchblende occurs in the spar in pieces 2 to 3 inches in diameter. These mines are worked principally for bismuth ocher, but also for cobalt and nickel.

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[a] Beck, Richard, Lehre von den Erzlagerstaetten: vol. 1, 1909, pp. 408–410.

In the quartzic cobalt-bismuth mines of Schneeberg, Saxony, are found bismutite and various minerals of nickel, silver, and arsenic. There is also some pitchblende, uranochalcite, troegerite, walpurgite, zeunerite, uranospinite, galenite, zincblende, and some copper compounds.

URANIUM DEPOSITS IN PORTUGAL.

The following description of the uranium deposits in Portugal is given by Segaud and Humery:[a]

> The uranium-bearing zone lies in the area of massive granite that occupies nearly the entire northern part of Portugal between the Desert of Galice to Càstello Branco and reaches into the provinces of Minho, Tras-os-Montes, and Beira. The richest part of the district is between the towns of Guarda and Sabugal, near the southern part of the granite area. The zone touches the outcrops of Cambrian rocks north of Guarda, the veins being especially rich in wolframite. The region of Villar-Formosa, however, is almost equally uraniferous.
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> This zone appears also south of Guarda, where the granite forms a superficial mass thrown above the primary rocks by faulting. Apart from the uranium, the rocks of the region are much mineralized, showing deposits of galena, arsenopyrite, chalcopyrite, tungsten, and cassiterite. The uranium-bearing veins crop out in the granite, and also in the Cambrian schists. The granite is generally hard, firm, and little altered. In the vicinity of the veins it is altered, perhaps through mechanical action. The degree of alteration varies much at different points. In some places a remarkable parallelism of the veins in one or two directions is observed; in one instance the veins can be followed 8 miles, and in another, less important, for a considerably longer distance. The width of the veins varies greatly. Outcrops show a continuous succession of swellings and constrictions, and the vein in many places disappears completely, only to reappear several yards farther away. In the north of Belmonte a vein is 8 meters wide for a short distance, but such width is believed to be exceptional. A width of from ½ to 1 meter is more common. The filling of the veins is mostly pegmatite and aggregations of crystallized feldspar and quartz. In the district of Guarda the filling of several veins is largely quartz. The gouge is argillaceous matter, and may carry uranium. Autunite (uranium-calcium phosphate) is present in different forms, in small groups of square tables of an intense yellow color, in small plates, as a pure coating, or still oftener in bright yellow specks disseminated through-

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[a] Segaud, —, and Humery, —, Deposits of uranium in Portugal: Ann. des Mines, ser. 11, vol. 3, February, 1913, pp. 111-119.

out rock of a dull yellowish color. In the clay part of the veins the uraninite is in places completely invisible and its presence is revealed only by the electroscope. There are, also, blotches on the surface of the granite, giving it an intense yellow color. Notwithstanding this color, the granite is always poor in uranium. The autunite is probably accompanied by uranyl-circite. Chalcolite of a beautiful emerald-green color is also present. The content of the veins varies enormously. At one place on an outcrop material was found carrying values of 4 to 5 per cent,[a] whereas the surrounding material carried no values whatever. Ore containing 2 per cent is excellent and that carrying 1 per cent is good average ore. The minimum limit for profitable exploitation is a content of 0.3 per cent to 0.5 per cent ore. The unaltered mineral has probably not yet been reached, except perhaps in a few places, and it is presumed that the mineral does not differ greatly from that in outcrops.

The region of Guarda forms an elevated plateau having an altitude of about 2,700 feet. It is traversed by a railway from Pampilhosa to Villar-Formosa and from Guarda to Lisbon. The transportation to the railroad stations is by oxcarts, but the erection of an aerial tramway is possible. The main work has been done in the more important districts of the northeastern part of the region around Porto. The region contains numerous small hamlets from which laborers can be easily procured at small wages.

The most important workings are those of the Compagnie de l'Urane, which employs 600 men. The average net cost per yard of drifts with a cross section of 3½ by 4½ feet is about $17, and the average net cost of 1 cubic meter of broken ore is about $20. The small mineral content of the ore makes treatment at the mine necessary. This treatment is entirely chemical, without previous mechanical concentration; the products are uranium oxide on one hand and sulphate of barium, rich in radium, on the other. The most important factory is at Baraçao, and is owned by the Compagnie de l'Urane. The radioactive barium sulphate is treated in the factories of Nogent.

URANIUM ORES IN AUSTRALIA.

Within the last few years several finds of uranium-bearing ores have been reported from Australia. These reports, apparently, have interested not only scientific men in Australia, but also the general public. One of these deposits is 80 miles east of Farina, a railroad station on the Great Northern line in South Australia, and lies between Mount Painter and Mount Pitt. Brown[b] states that the rocks of the district consist of coarse and fine feldspathic, siliceous, and micaceous granite, and gneiss, and also micaceous rocks, quartzite, and mica schist. The rocks are contorted in places and penetrated by dikes of coarse, pink colored, eruptive granite.

Two of the prospect pits are on outcrops of iron oxide with cellular quartz and gossan, the whole having the appearance of an irregular lode. The uranium minerals, torbernite and autunite, are disseminated through the ore, and are also crystallized on the walls of the fissures and cavities in it. Uranophane and gummite occur sparingly. Fergusonite and some monazite are also present. Torbernite occurs on two of the other claims, and on still another both torbernite and autunite are disseminated through the rock and in seams. The extent and width of the ore deposit have not been determined.

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[a] The article does not state what these percentages indicate. They probably refer to U_3O_8.

[b] Brown, H. L. Y., Uranium ores in South Australia, 1911, p. 3.

Another uranium deposit lies southeast of the one just described, about 20 miles east-southeast of Olary, on the railroad line from Petersburg to Broken Hill, South Australia. The ore occurs as a yellow and greenish-yellow incrustation and powder on the faces, joints, and cavities of a lode, which consists of titaniferous magnetite, magnetite, etc., and quartz in association with black mica. There are two of these lode outcrops, more or less parallel and 5 to 15 yards apart. The main outcrop can be traced for some 200 yards. The ore is reported to consist almost entirely of carnotite, with possibly some gummite.

From the accounts published most of the ore is of a very low grade. A company has been formed for exploiting the ore and extracting the radium.

VANADIUM FROM ORES OTHER THAN CARNOTITE.

SAN MIGUEL COUNTY, COLO., DEPOSITS.

Probably the largest deposits of vanadium that have yet been discovered in the United States are in southwestern Colorado between and close to Placerville and Newmire in San Miguel County. These deposits were visited by Ransome and Spencer in 1899 and their description, together with notes on the chemical analyses and composition of roscoelite by Hillebrand, was published in 1900.[a] Fleck and French have also described the deposits.[b] Fleck and Haldane later published additional descriptions, with notes on mining operations.[c] Hess, in 1912, published an excellent description of these deposits with notes on the possible origin, etc.[d]

According to Cross and Purrington,[e] the country rock is composed of Jurassic and Triassic sediments. They have divided these into three formations, the Dolores below, La Plata above, and McElmo above the La Plata. The latter is composed of two heavy beds of light-colored sandstone, separated by a thin bed of limestone. The vanadium-bearing rock is the lower sandstone which probably consists of two beds with an unconformity between. That part above the unconformity is yellowish in color and that below is gray.

These beds outcrop on both sides of Bear Creek, south of Newmire, at a height of about 50 feet above the creek bed. A chemical company with works at Newmire, Colo., and Primos, Pa., is mining these Bear Creek deposits. On the east side of the creek the workings are rather extensive. The ore is removed as found, although such operations have resulted in the formation of large chambers. The mining is easy, as the sandstone roof is hard and easily supported. Beyond these chambers are several drifts. Mining at present is confined to the west side of the creek. There is an easy haul of about 2½ miles from the workings to the plant, with a down grade all the way.

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[a] Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: Am. Jour. Sci. 4th series, 1900, vol. 10, pp. 124–144; U. S. Geol. Survey Bull. 262, 1905, pp. 10–14, 18–22.

[b] Fleck, H., and French, S. W., Uranium and Vanadium: Quart. Colorado School of Mines, January, 1909.

[c] Fleck, H., and Haldane, W. G., Preliminary report on the radioactivity of carnotite in southwestern Colorado: Quart. Colorado School of Mines, October, 1909.

[d] Hess, F. L., Notes on the vanadium deposits near Placerville, Colo.: U. S. Geol. Survey Bull. 530, pt. 1, 1911, pp. 142–157.

[e] Telluride Folio No. 57, Geol. Atlas, U. S., U. S. Geol. Survey, 1899, p. 52.

The vanadium-bearing rock is a light to dull green fine-grained sandstone. Sometimes the color is quite dark. Occasionally splotches of carnotite are found in the cracks and fissures, but the uranium content is too small to be worth saving. As a rule the richest ore follows the apparent unconformity between the two sandstones. In places a shallow layer, dark in color and from one-half to one inch thick, lies close to the unconformity and is said to be rich in vanadium. This layer partly, at least, accounts for the origin of the vanadium in the sandstone above and below.

The deposits are also found at Sawpit between Newmire and Placerville, on both sides of the Rio Grande River; in the canyon of Fall Creek which runs into the Rio Grande below Sawpit; on the east side of the Rio Grande both north and south of Placerville; and on both sides of Leopard Creek, which runs into the Rio Grande close to Placerville. The only ore being mined at present, however, is at Bear Creek. According to Hillebrand,[a] the green vanadium mineral to which the sandstone owes its color is not a chlorite notwithstanding its appearance, but is closely related to the mica roscoelite wherein the proportions of Al_2O_3 and V_2O_3 are reversed. A large proportion of the ore carries less than 1 per cent V_2O_5. The ore mined at Bear Creek has an average content of 1½ per cent V_2O_5. Some of it contains as much as 2 per cent V_2O_5, or even a little more. The thin layer, already referred to, at places carries more than 8 per cent V_2O_5. These low-grade roscoelite deposits can be mined at a profit, because they are large and easily worked.

HUERFANO COUNTY, COLO., DEPOSITS.

Vanadium ore has been recently discovered in Huerfano County, Colo. The Colorado Mining Corporation claims, six in number, are in the Culebra spur of the Sangre de Cristo Range in Huerfano County. The nearest railroad station is Russell Siding, about 9 miles distant. From this siding to within a short distance of the mines the wagon road is an abandoned railroad grade. The vein is said to be a well-defined fissure vein and has been opened up at different places on the surface for a distance of 2,500 to 3,000 feet. It is 1 to 4 feet in width. The lowest workings are about 25 feet deep with the best showing in a couple of shafts about 14 feet in depth. The vein was originally worked for copper, but about a year ago it was found to contain vanadium. A number of assays show the following V_2O_5 content: 2.63, 2.91, 6.25, and 7.35 per cent. The copper content is 2 to 4 per cent or a little more. One assay made upon a much larger sample than was used in any of the above determinations showed 4.5 per cent V_2O_5. The ore is heavy, black, and banded and is probably gneiss impregnated with vanadium minerals. It carries much green material, some of which may be copper vanadate. In places the ore is rather yellow and assays are reported showing as much as 1.75 per cent U_3O_8, but none of the yellow samples we examined contained more than a trace of uranium. Two or three small outcrops carrying vanadium have been found within a mile and a half east of these claims. Five miles north is a thin vein from 1 to 2 inches thick called the Santa Rosa vein, which also is said to carry some vanadium.

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[a] Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, p. 21.

CUTTER, N. MEX., DEPOSITS.

At Cutter in Sierra County, N. Mex., on the Atchison, Topeka & Santa Fe Railroad, deposits of vanadinite have been worked by the Vanadium Mines Co. The veins lie about three-fourths of a mile south of Palomas Gap and contain, in addition to vanadium, galenite, copper carbonates, barite, fluorite, and other minerals. They were visited by Hess in 1911.[a] The mine has recently been abandoned and the plant removed.

Some ore has been mined on the Widner claims during the past year, but it was shipped for the lead content only.

EAGLE COUNTY, COLO., DEPOSITS.

In Eagle County, Colo., 7 miles southeast of the town of Eagle, silver ore has been found that carries also vanadium. The Lady Belle mine is 400 feet above Brush Creek on the side of Horse Mountain. There are two tunnels, one about 25 feet above the other. The upper one runs about north and follows the strike of the ore body, the dip of which is 34° E. The lower tunnel runs at an angle to the upper one and then bends until it takes almost the same direction. It is in about 60 feet and the upper one about 40 feet. The ore is a dark-greenish sandstone similar in appearance to the darker types of roscoelite found at Newmire, Colo. It assays 25 to 1,000 ounces of silver to the ton. Much of the ore that is high in vanadium is low in silver, although this is not invariably true. The vanadium ore contains coarsely crystalline layers. The high-grade silver ore is more compact, usually darker, and has blue-black dots and splotches, due probably to silver bromide. Pieces of float ore rich in silver have been found near Horse Mountain, but no high-grade ore has been found in place at the time of writing, except at the Lady Belle mine.

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[a] Hess, F. L., Notes on the vanadium deposits near Placerville, Colo.: U. S. Geol. Survey Bull. 530, pt. 1, 1911, pp. 157-160.

A section across the ore body, beginning with the foot wall, is as follows: Four feet of rich ore (A), 11 feet of low-grade ore (B), 4 feet of rich ore (C), and — feet of low-grade ore (D). The hanging wall has not yet been reached, and therefore the total thickness of the ore body is not known.

Vanadium assays made by the Bureau of Mines show the following results:

Vanadium content of ore from Lady Belle mine.

Part of section of ore body.	V_2O_5 content. *Per cent.*
A (lower part)	2.66
A (upper part)	.21
B	2.30
C	1.02
D	.38

A sample across the breast of the lower drift showed 0.94 per cent of V_2O_5. No attempt to recover the vanadium has been made.

DEPOSITS IN OTHER STATES.

A deposit of vanadium ore has recently been discovered in California. It is on low ground, a few rods from a good road, 5 miles from Klinefelter Station, on the main line of the Santa Fe Railroad, near the eastern border of San Bernardino County. It is stated that the vein is 8 feet wide and can be traced on the surface for a distance of 435 feet. The ore is largely calcite. One sample tested by the Bureau of Mines showed a content of 1.71 per cent V_2O_5. Water is near by, there being numerous springs.

There are several deposits of grahamite in the United States, those in West Virginia, Oklahoma, and Nevada probably being the most important. Grahamite is a solid native bitumen, the origin of which was first described by White[a] as being derived from the oxidation of petroleum. Richardson[b] mentions the fact that both the West Virginia and Oklahoma grahamites contain vanadium in the ash. The mineral has been described as a brittle, solid, native bitumen, the result of the metamorphism of petroleum, generally pure, but at times containing adventitious mineral matter. Grahamite does not melt, but glows and burns slowly on the application of heat.

The deposits near Page, Le Flore County, Okla., and in the Impson Valley, are fully described by Taff.[c] At Page very little development work has been done. An adit has been driven in the vein and within the adit a shaft has been sunk in the deposit, following the vein for a considerable depth. The ore outcrops several hundred feet higher up on the hillside. The property has been leased to a Pittsburgh concern for the last few years, but the lease does not call for the working of the property and it has been idle during this time.

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[a] White, I. C., Origin of grahamite: Bull. Geol. Soc. of America, 1899, vol. 10, pp. 277-284.

[b] Richardson, Clifford, Grahamite, a solid native bitumen: Jour. Am. Chem. Soc., vol. 32, pt. 2, 1910, pp. 1033-1049.

[c] Taff, J. A., Grahamite deposits of southeastern Oklahoma: U. S. Geol. Survey Bull. 380, 1908, pp. 286-297.

The Oklahoma grahamite burns with a smoky flame to a yellowish-brown ash, whereas the grahamite from West Virginia, under similar conditions, forms a pasty asphaltic mass when the volatile matter is driven off, and finally reduces to a reddish-brown ash.

Samples analyzed by the Bureau of Mines laboratory at Denver, Colo., gave the following results:

Analysis of grahamite for vanadium.

	Grahamite from West Virginia.	Grahamite from Le Flore County, Okla.	
		From drift.	From outcrop.
	Per cent.	*Per cent.*	*Per cent.*
Ash	2.11	0.83	3.39
V_2O_5 content in ore	.17	.28	.22
V_2O_5 content in ash	7.94	34.5	6.77

DEPOSITS OF PATRONITE IN PERU.

The vanadium ores from the mines in the United States meet a strong competitor in the patronite[a] shipped from the deposits in Peru owned by the American Vanadium Co., of Pittsburgh, Pa.

These deposits are at Minasragra, 20 miles from Cerro de Pasco. The area lies along the western limit of a broad anticline in "Jura-trias" and Cretaceous rocks. A section shows the series in this locality to be composed of green shales, thin beds of limestone, and red shales. Vanadium is found only in the red shales. The deposit proper appears to be a lens-shaped mass, 28 feet wide and 350 feet long. The dip is 75° W., and the strike is N. 20° W. The ore contains several minerals. The mineral that constitutes the larger portion of the deposit has been called "quisqueite." It is a black carbonaceous substance containing sulphur, with a hardness of 2.5 and a specific gravity of 1.75. There is also a lesser quantity of a cokelike material with a hardness of 4.5 and a specific gravity of 2.2. Neither of these contain vanadium. The vanadium is mostly at the southern end of the ore body, and to a depth of 20 feet is largely in the form of red calcium vanadate. The color is brighter than that of the calcium vanadate found in Colorado and Utah, and the ore carries as much as 50 per cent vanadium oxide. It occurs in small pockets and fills the cracks and fissures in a fine shale. Below this shale is the "mother lode." It is 9 to 30 feet thick, extends along the greater length of the deposit, and dips 40°. It carries as high as 10 per cent vanadium oxide and nearly as much sulphur. On the east and south sides, below the "mother lode," is found a hard blue-black vanadium shale, carrying as much as 13 per cent vanadium oxide and 4 to 5 per cent sulphur. Patronite, the main vanadium mineral, is greenish black and has a hardness of 2.5 and a specific gravity of 2.71. It contains from 19 to 24.8 per cent vanadium oxide and sometimes 50 to 55 per cent of combined sulphur. The patronite originally almost reached the surface close to a dike on the east side and the vein was followed in sinking the shaft. It is most abundant in the north half of the lens. The whole ore body is almost completely inclosed by porphyry dikes. There are also two or three intrusions in the ore body.

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[a] Hewett, D. F., A new occurrence of vanadium in Peru: Eng. and Min. Jour., vol. 82, Sept. 1, 1906, p. 385; Hillebrand, W. F., The vanadium sulphide, patronite, and its numerous associates from Minasragra. Peru: Jour. Am. Chem. Soc., vol. 29, pt. 2, 1907, pp. 1019–1029; Bravo, J. J., El Vanadio de Minasragra: Inform. Mem. Soc. Eng. Lima, 1906, pp. 171–185; Hewett, D. F., Vanadium deposits in Peru: Trans. Am. Inst. Min. Eng., vol. 40, 1909, p. 291.

PRODUCTION OF URANIUM, VANADIUM, AND RADIUM.

During the year 1912, 28.8 tons of uranium oxide, equivalent to 24.4 tons of metallic uranium, was produced in this country. In addition, 1.4 tons of uranium oxide was shipped, but has been held up in transit because the uranium oxide content was so low that it could not be marketed. The value of the uranium content of the ore shipped, on a basis of $1.50 per pound, is $86,000. On the basis that the radium is in equilibrium (see p. 66) with the uranium, 9.77 grams of radium chloride, or 12.7 grams radium bromide (anhydrous), were contained in the uranium ores mined and shipped in this country during 1912. All but a few tons of these ores, as already stated, was sent abroad and the radium was extracted in Europe. It is difficult to say at this time what the exact ratio of radium to uranium in carnotite ore is, as very little work has been done on the subject except in one commercial laboratory. Private information from this laboratory indicates that the ratio is only a little below the normal ratio of 1 part radium to 2,940,000 parts uranium. Allowing a generous margin of 10 per cent, the actual production of radium from American ores last year, assuming that all was extracted, was equivalent to 8.8 grams of radium chloride, or 11.43 grams of the bromide, worth, at the present price of $90,000 per gram for chloride, $792,000.[a]

We have been unable to ascertain the exact production of radium from other sources during the year 1912, as the figures from the Austrian mines are not yet available. The total production of radium preparations from the Austrian mines in 1911, calculated as pure radium chloride, was 2.647 grams, valued at $211,750. The production of radium from other uranium ores mined in 1912, omitting Austria and the United States, is probably less than 1½ grams of radium chloride. The total production of radium chloride from foreign ores, therefore, during the year 1912, assuming that the Austrian production was no larger in that year than in 1911, was less than 4 grams. Therefore American ores supplied more than twice as much radium as was obtained from all other sources.

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[a] Some of the ore shipped toward the end of 1912 did not reach Europe until after the close of the year, and the radium in it would probably not be extracted from it for several months. Although the production of carnotite ore in the United States in 1911 was only a little less than that of 1912, less of it was sent abroad and what was shipped at the end of 1911 would not quite offset what was shipped at the end of 1912, so the above figures on production of radium in 1912 are a little high, although it represents the radium in the ore shipped.

American ores in 1912 supplied approximately 285 tons of vanadium metal in the form of ferrovanadium and other vanadium products. Some of this went abroad, but most of it was used in this country.

USES OF VANADIUM, URANIUM, AND RADIUM.

USES OF VANADIUM.

The main use of vanadium is as an alloy in steels where great toughness and torsional strength are required, such as automobile parts, gears, piston rods, tubes, boiler plates, tires, transmission shafts, bolts, gun barrels, gun shields, and forgings of any kind which have to withstand heavy wear and tear. The vanadium content in such steels varies from 0.1 to 0.4 per cent. It is occasionally used in certain tungsten alloys for making high-speed tool steel. The introduction of a small proportion of vanadium decidedly reduces the proportion of tungsten required to give such alloys the desired hardness and toughness.

Arnold[a] has given some illustrations of the effect of vanadium upon steels of different types:

> One plain carbon steel containing about 1 per cent of carbon had a yield point of 35 tons per square inch, a maximum stress of 60 tons per square inch, an elongation of 10 per cent on 2 inches, and a reduction of area of 10 per cent. The addition to this steel of about 0.6 per cent of vanadium raised the yield point from 35 to 65 tons, the maximum stress from 60 to 86 tons per square inch, still leaving an elongation of 7 per cent and a reduction of area of 8 per cent.
>
> A steel containing 0.25 per cent of carbon and 3.3 per cent of nickel gave a yield point of 33 tons, a maximum stress of 42 tons per square inch, an elongation of 26 per cent on 2 inches, and a reduction of area of 53 per cent. A practically identical steel, but containing in addition about 0.25 per cent of vanadium, gave a yield point of 50 tons instead of 33, a maximum stress of 68 instead of 42 tons per square inch. The elongation was 17 per cent on 2 inches and the reduction of area 36 per cent.
>
> A steel containing 0.25 per cent of carbon and about 1 per cent of chromium registered a yield point of 27 tons and a maximum stress of 41 tons per square inch, with an elongation of 36 per cent on 2 inches and a reduction of area of 55 per cent. The addition of 0.25 per cent of vanadium raised the yield point from 27 to 40 and the maximum stress from 41 to 55 tons per square inch. The elongation was lowered from 36 to 26 per cent and the reduction of area from 55 to 53 per cent.

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[a] Arnold, J. O., Some recent advances in scientific steel metallurgy: Nature, March 20, 1913, p. 70.

Vanadium, therefore, differs from tungsten in having an extremely beneficial effect not only on tool but also on structural steel. Arnold has shown that vanadium seemingly does not form a double carbide with iron, but gradually takes the carbon from the carbide of iron until, if about 5 per cent of vanadium is present, Fe_3C can not exist, and only a vanadium carbide, V_4C_3, containing 15 per cent of carbon, is present, and this constituent is constant, at least in tool steels containing 5 to 14 per cent of vanadium. The micrographic analysis of such alloys has resulted in the discovery of three new constituents, namely, vanadium pearlite, vanadium hardenite, and vanadium cementite.

There seems to be a tendency to substitute the use of titanium to some extent for that of vanadium, although titanium probably acts only as a reducing agent. Vanadium is also used in making bronzes. in medicine, and in dyeing.

USES OF URANIUM.

Uranium salts have been used for many years in glass manufacturing. Uranium colors glass yellow, and in sufficient proportion imparts to glass a beautiful fluorescent color known as "opalescent." Fifteen per cent or more of the oxide may be required to give the desired effect. It is also used in ceramics for the purpose of obtaining brilliant, fireproof tints of yellow, orange, and black. Uranium coloring powders may be obtained in black or in six shades of yellow.

Uranium can be used as an alloy of steel, but alloys of other metals that have similar properties can be produced more cheaply. Owing to the increased supply of uranium, however, experiments are once more being tried with the object of getting some alloy with properties of a sufficiently distinctive character to make it a commercial product.

USES OF RADIUM.

Radium is used in scientific research and in medicine. A study of radium and its disintegration products has vastly extended the conception of the composition of matter and the nature of the elements. Owing to the cost of the material, however, the quantity available for scientific research must necessarily be limited, although it is unfortunate that a larger proportion of the radium supply can not be devoted to purely scientific purposes. The commercial demand for radium must depend largely upon what use can be made of it for medical purposes. The following data have been abstracted from scientific journals and in part obtained from the bulletin published by the Imperial Department of Public Works in Austria:

Radium treatment is given by means of baths in radioactive waters, by drinking radioactive waters, by subjecting the patients to the radium emanation, by doses of the radium emanation artificially dissolved in water to a much greater strength than can be obtained from the natural waters, by subcutaneous injections of water containing the emanation in solution, and by direct exposure to the radium rays from radium preparations.

A great number of natural springs have been found to be radioactive. In fact, practically all natural waters contain at least small quantities of the radium emanation in solution, although comparatively few contain radium salts in solution. To say that a water is radioactive, therefore, means very little, as any deep-well water is to some extent radioactive, and even river waters are often slightly so. The important point is not that the water is radioactive, but to what degree it is radioactive. The variation in radioactivity is marked, some springs being hundreds and even thousands of times as radioactive as others. The radioactivity is, of course, acquired by underground water coming in direct contact either with radium emanation which the water dissolves or with radium-bearing ores. Many well-known mineral springs are strongly radioactive, and there has been a tendency to attribute at least some of their therapeutic value to this radioactivity.

Recently the Austrian Government has established baths in connection with the mines at St. Joachimstahl. It is claimed that the waters from the mines have a special value in the treatment of subacute, and especially of chronic, rheumatism of the joints and muscles, for gout, diabetes, and various forms of neuralgia of a rheumatic or gouty nature. It is also claimed that the waters are beneficial in the cases of chronic catarrh of the mucous membranes, slight paralysis, anemia, chlorosis, lymphatic disorders, and diseases of women. The treatment is not confined to baths only, but all of the methods indicated above are used. Direct exposure to radium rays is especially recommended for rheumatism, neuralgia, stiffness of the bones and joints, as well as chronic eczema.

An abstract of part of the recent report of the Radium Institute of London follows:

It is necessary to get a constant source of radiation in the treatment of diseases by means of radium. The different forms of apparatus used for this purpose are called "applicators," and may be of two kinds (*a*) those containing radium salts, and (*b*) those containing the emanation or gas evolved by radium salts. The first may be divided into two types, (1) in which the radium salts are fixed by a varnish, and (2) tubes containing the radium salts.

In order to standardize these instruments a unit of concentration has been adopted. It is represented by 1 centigram of radium bromide covering 1 square centimeter of surface. Such applicators are termed "full strength." One with half this quantity on the same area is termed "half strength," and so on. In order to compare instruments with applicators used elsewhere, in future the instruments will be standardized in terms of metallic radium rather than radium bromide.

The varnished applicators are made in the form of square, rectangular, and circular plates of silver. Radium sulphate is mixed with varnish and spread over the metal and, after drying, three individual coats of varnish are laid on top. The whole is finished with a last coat of a particular type of varnish which gives a glasslike surface. Such instruments can be sterilized by means of heating to 120° C., or by means of alcohol, mercuric chloride, or ether. It is recommended that the tube applicators be packed tightly so that the radium salts can not be shaken around inside. The emanation applicators are either flat or in the form of tubes. The flat instruments are small, hollow boxes made of German silver with one face turned down in a lathe to a thickness of 3 mm. The instrument is fitted with a lead tube with a capillary bore. The mixture of oxygen and hydrogen which is obtained from solutions containing radium salts is pumped off by means of a vacuum pump with the contained emanation. On exploding the mixed gases a small quantity of hydrogen is left containing all of the original emanation. The applicator is then partly exhausted, the hydrogen and the emanation allowed to run in and the end of the capillary tube pinched, conveniently retaining the gases inside.

The tube applicators have a volume of 2 c. c. or more, and are filled with hydrogen containing emanation obtained in the manner already described. If smaller tubes are desired, the emanation is frozen out by means of liquid air and the hydrogen pumped off. The tube can then be sealed.

The institute has also prepared a large quantity of radioactive water for its use, such water having a strength of from 1 to 2 millicuries[a] per liter, which is much stronger than even the more radioactive natural waters.

From August 14, 1911, to December 31, 1912, a large number of patients were treated. There seems to have been no disposition to select cases in any way. The general report on these cases is as follows:

Report on patients treated from Aug. 14, 1911, to Dec. 31, 1912.

Examined but not treated	38
Received prophylactic irradiation only	39
Apparently cured	53
Cured	28
Improved	245
Not improved	70
Abandoned treatment	88
Dead	55
Recently treated and results not yet noted	41

These cases covered a large number of diseases, including carcinomata of the larynx, uterus, neck, etc., rodent ulcer, malignant diseases of skin, parotid tumor, adenoma of breast, adenoma of thyroid, fibroid diseases of uterus, leucoplakia, tropical ulcer, various skin diseases, locomotor ataxia, diabetes mellitus, etc.

Statements from another source indicate that 80 per cent of cases of arthritis treated with comparatively strong doses of the radium emanation dissolved in water are either cured or greatly benefited. A similar treatment seems to be beneficial for rheumatism. The dosage in such cases is as follows: The radium emanation obtained in one week from 1 gram of radium bromide in solution is dissolved in 17 liters of water. One hundred cubic centimeters of this solution is taken twice a week by the patient. This means that 1 gram of radium bromide can be used for treating about 170 patients at one time.

A word of warning seems not out of place here. There has already been a disposition to exploit so-called radioactive waters whose radioactivity is no larger than that of ordinary deep-well water. It is by no means an assured fact that even those waters which have a radioactivity somewhat above the average can claim any special healing qualities from their small radium content. It is open to doubt whether cures that have been effected are due to the small proportion of radium, or to other substances in the water. A spring at Seligman, Mo., somewhat largely advertised, was tested by a well-known chemist and proved to have a radioactivity of 3.5 mache units—about the same as deep-well water. The exploitation of radium in medicine has been carried even further. Inhalers, which usually involve the inhaling of oxygen also, and "bath salts," containing small quantities of radium, are on the market, and the number is likely to increase. It is quite possible that some of these may have a use, but their value has not by any means been thoroughly demonstrated. It must be borne in mind that proper treatment, involving the use of radium, can not be given at present by the average physician, at least to the best advantage, because he is not well informed as to the properties of the substances used. For example, the radium emanation loses half its strength in 3.8 days. That is to say, after 3.8 days half of the gas has changed into disintegration products. At the end of one month all of it has changed and there is no emanation left. Without some knowledge of radioactivity, a physician would frequently not be able to tell the strength of the material he was using.

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[a] One millicurie is the quantity of emanation in equilibrium with 1 mg. of radium.

Although the medical value of radium has been under test for several years, the quantity available for such experiments has been limited. Now that the supply of radium salts has increased a greater proportion of this supply ought to be devoted to scientific research in order that knowledge in regard to its uses in the arts and its medicinal properties may be extended.

RADIUM INSTITUTES.

As already stated, only a comparatively small part of the total supply of radium is at present available for use in medical and other scientific research.

The Radium Institute of Vienna was the donation of Dr. Kuppelweiser, a philanthropist of Vienna, to the Academy of Sciences of Austria, which has turned over the direction of the institute to the department of physics of the University of Vienna. The institute is purely a research institution, offering no courses of instruction and accepting only investigators of recognized standing. It owns about 2½ grams of relatively pure radium salts presented to it by the Austrian Government. About 1 gram of this may be regarded as 100 per cent pure. Although the institute has neither the right to sell nor loan radium, it is the repository for such preparations as the Government is holding for sale and can use them as a source of radiation for experimental purposes. Sales of radium by the Government are made on the basis of the measure of radioactivity of the substance as determined by the institute. The institute has confined its research work to purely scientific investigations and has rather systematically avoided connection with medical researches.

In Paris the laboratory of Madam Curie is a part of the Sorbonne, or University of Paris, although not at present in the Sorbonne building. It will soon be moved to the new Radium Institute which the university has built near the Pantheon. The radium owned by the laboratory was orignially presented to Madam Curie in the form of uranium residues. Only about 1 gram of radium has been extracted in any degree of purity, but a large quantity of residue still remains on hand. This gram of radium is used for purely scientific research and not for medical purposes.

The English Radium Institute was founded by Sir Ernest Cassel and Viscount Iveagh, who gave a large sum for its endowment. The institute does not own a mine nor extract radium from ore, but buys whatever refined radium salts it needs. It has at its disposal probably somewhere between 1 and 2 grams of radium chloride. Its work is entirely confined to the medical and other scientific uses of radium, primarily the former. The annual report, which has recently been issued, has already been mentioned.

MARKET VALUE OF RADIUM SALTS.

The price of radium salts varies to some extent, depending upon the quantity purchased and also from whom purchased. The average price for radium bromide in small quantities is $70 per milligram. Some makers charge as high as $100 per milligram. This, however, is for a specially pure and guaranteed product. Lower-grade material has a correspondingly lower price. For example, a sample containing 50 per cent radium bromide is worth half as much per milligram as one which is guaranteed to be perfectly pure. A price of $70 per milligram corresponds to $70,000 per gram, or $2,000,000 per ounce.

There is hardly any material on the open market to-day that offers better chances for fraud in connection with its sale. The reason of this is that the average chemist is unable to tell whether the radium salt is pure, and to what degree it is below the guaranteed standard. In the earlier days of the sale of radium a radioactivity standard was used. Under this standard any material that had a radioactivity of two million[a] was represented as pure. Material of lower radioactivity contained a proportionate amount of radium. The standard was absolutely arbitrary and for temporary purposes was sufficiently exact, but conditions have changed. A person purchasing radium should insist on receiving a guarantee, stating exactly what per cent of radium chloride or radium bromide the sample contains. Upon no other basis can a purchaser be sure of obtaining a standard article.

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[a] The radium was supposed to be 2,000,000 times as radioactive as a similar weight of pure uranium oxide.

Another point which it is necessary to bear in mind is that in radium chloride and radium bromide the percentage of metallic radium is very different, the chloride containing 76 per cent radium and the bromide less than 59 per cent. It can therefore be seen that the values of a milligram each of the two salts, based upon the metallic radium content, are different. The same price per milligram should not be paid for bromide as for chloride. All dealers should sell their products on the basis of the proportion of metallic radium present.

With the large increase in production of radium during the last two years, the question of a market and the continuance of the present prices is a very pertinent subject. Undoubtedly it is now more difficult to dispose of radium than it was a few years ago, due to this increase in production. As already stated, the future market will depend very largely upon the successful use of radium for medical purposes. If it can be demonstrated to the satisfaction of the medical profession as a whole that it has a decided use in any one disease, the demand for radium will not only equal the present supply but will equal any supply that is likely to be put upon the market in the future. For example, if it can be proven that treatment with the radium emanation is the best cure for rheumatism, since 1 gram of radium chloride is enough for treating only 170 persons continuously, it can be seen that no likely supply in the immediate future could equal the demand. Therefore a supply of radium salts sufficient to meet the needs of an adequate investigation of the medicinal properties of radium should either be placed in the care of a radium institute in this country, or made available for use in some of our large hospitals. The radium in the Colorado and Utah ores should not be sent abroad in the future as it has been in the past. This country should receive the scientific and medical benefits which may be expected from its retention and use on this side of the Atlantic.

RADIOACTIVE METHODS FOR TESTING ORES.

The matter of sorting uranium ores correctly is most important to the practical miner. From the descriptions of the ore bodies in Paradox Valley and in Utah, one can readily see that it is sometimes extremely difficult to tell what is and what is not a shipping ore. The fact that in many cases long hauls are necessary to carry the ore to the railroad makes it doubly important that no ore should be shipped that can not be marketed. The miners learn by experience to distinguish high-grade from low-grade ore, but as the appearance of the ore varies in different localities, a sorter who is successful in one place may make serious errors in another. In addition, as already stated, a number of analytical mistakes have been made by some assayers with rather serious consequences to some of the shippers. It therefore becomes exceedingly important to have some quick method by which a mine operator can tell whether the uranium oxide content of his ore is or is not above the minimum of 2 per cent. This can be readily accomplished by an intelligent person with a little practice by means of an electroscope.

METHOD OF USING THE ELECTROSCOPE FOR APPROXIMATE DETERMINATIONS.

A suitable electroscope usually consists of two compartments; one above containing a suspended gold leaf in front of which is attached a reading microscope, and one below in which the ore to be tested is placed. Usually the leaf is electrically charged by means of a piece of vulcanite rubbed on the sleeve of the coat, the charge causing the leaf to rise; then the natural leak of electricity from the leaf is noted on the scale, and calculated as a certain number of divisions per minute. The ore is then placed in the compartment below and the leak of the leaf noted as before. If the ore contains uranium and radium, the rate at which the leaf falls will always be faster than the natural leak of the instrument itself. This is due to the fact that the rays given off by the radioactive material ionize the air in the compartment in which it is placed, and if the leaf has been charged positively the negative ions will be attracted to the leaf system and will discharge the charge which has been imparted to it. This ability to discharge electricity is the means by which we recognize radioactive substances.

There are a numer of precautions, however, to be taken in making such measurements. First, the illumination during the taking of the readings should be constant. It is therefore much better to make all determinations at night, when an artificial light can be placed at a definite distance behind the electroscope. Second, readings should always be taken between the same points on the scale. Third, in comparing two ores their physical condition should be as nearly as possible the same. This may be roughly done by passing them through the same mesh sieve, preferably 40 or 60 mesh. Of course, every particle of the ore must be ground until it finally passes this sieve, otherwise there will be a partial concentration of the radioactive material in the finer portion. The same weight should be taken and the same surface should be exposed in the electroscope. A balance is inconvenient in a mining camp, but approximately uniform quantities of ore may be obtained in the following way:

In a brass plate about one-fourth inch thick, of a size to fit into the bottom compartment of the electroscope, should be cut by means of a lathe a circular depression one-eighth inch deep and about 2 or 3 inches in diameter. This can be done by any brass worker. The bottom and sides of the depression should be perfectly smooth. The ore to be tested is poured into the depression, the plate tapped gently so as to settle the ore, and then by passing the edge of a flat piece of metal across the surface of the plate the extra ore is wiped off and the depression left exactly filled with ore with a perfectly flat surface.

In this manner a fairly uniform weight of material is obtained for comparison and the surface exposed in the electroscope is approximately constant. Of course, the density of the ores tested varies, but the method is accurate enough to give approximate results.

The plate with the ore is introduced into the bottom compartment of the electroscope and a reading taken. The ore is removed and replaced by a sample of carnotite of known uranium content which serves as a standard. This sample, of course, is passed through the same mesh sieve as the sample being tested. The relative radioactivities; that is, the rates at which the leaf falls, are roughly proportional to the amount of U_3O_8 present, it being assumed that the ratio of radium to uranium in the two ores is the same. The following example will show how to make a calculation:

Method of calculating percentage of U_3O_8.

Natural leak of instrument=5 divisions in 10 minutes.
Natural leak of instrument=0.5 division per minute.
Rate of fall of leaf with standard ore (3 per cent U_3O_8)=48.5 divisions per minute.
Rate of fall of leaf with ore to be tested=36.5 divisions per minute.

Subtracting from each of these figures the natural leak, 0.5 divisions per minute, give 48 and 36. The percentage of U_3O_8 in the ore will then be $\frac{36\times3}{48}=2.2$.

If the natural leak is as low as 0.5 division per minute and the radioactivities of the samples are as high as those indicated in the above experiment, the natural leak can be neglected, as the error from it is less than the probable experimental error.

The following actual comparison of the radioactivity of two carnotite ores, the uranium content of which was determined by analysis, will show the probable error in work that is carefully done.

Comparison of two carnotite ores.

	First ore.	Second ore.
Percentage of U_3O_8	2.02	3.16
Divisions per minute	7.4	12.0

If the 2.02 per cent ore be taken as the standard of comparison, the electroscopic results indicate that the second ore contains 3.27 per cent U_3O_8 instead of 3.16 per cent, the agreement being close enough for practical purposes.

The method outlined above is only approximately correct, because every sample of radioactive material has a different emanating power; that is, it occludes the emanation to a different degree. Other factors, as already explained, affect the result and are difficult to control.

However, with care and a little practice the electroscopic method can be successfully used to give an approximate indication of the uranium content of the ore. The total time required for a determination, including the grinding of the material, should not be more than one-half hour.

If the radium content of the ore is desired, in order to obtain an accurate determination, it is necessary to use proper, if small laboratory facilities, a different type of electroscope, and a little more skill in manipulation than is necessary for getting a rough approximation of the radioactivity by the method already described. There are a number of electroscopes on the market which can be used. A suitable one is shown in Plate IV, *B*. The upper portion, which contains the leaf system, can be unscrewed and removed from the lower portion, which allows repairs to the leaf system to be readily made. The insulation is either sulphur or amber. As the lower compartment must be absolutely airtight and it is difficult to get a satisfactory contact between the sulphur or amber and the brass, a cement should be used. One made by melting a mixture of rosin and black rubber is satisfactory. The lower compartment contains a thin brass plate or cylinder connected by means of a brass wire with the leaf system in the upper compartment. The charging device is plainly shown in the figure. The lower compartment can be partly exhausted and a radioactive gas run into it.

METHOD OF USING THE ELECTROSCOPE FOR EXACT DETERMINATIONS.

STANDARDIZATION OF THE ELECTROSCOPE.

The electroscope has to be standardized; that is, the effect on it of the emanation that is in equilibrim with a given quantity of radium must be determined. When this is known, the radioactivity of the gas can be calculated by comparison with the effect obtained from the standard radium emanation. The method of standardization is based upon the fact that in any unaltered uranium mineral the ratio of the radium present to uranium is constant. Rutherford and Boltwood[a] have determined this ratio and found that 1 gram of uranium is in radioactive equilibrium with 3.4×10^{-7} grams of radium. This means that a quantity of pitchblende, or any other unaltered uranium mineral, which contains 2,000 pounds of metallic uranium, or 2,360 pounds of uranium oxide, will have in it 308 mg. of metallic radium, which is equivalent to 404 mg. of radium chloride.[b] If, then, the emanation from a small weighed quantity of a uranium mineral, in which the percentage of uranium is known by chemical analysis, be introduced into the electroscope and the rate of fall of the leaf noted, a constant for the electroscope may be calculated from the data that shall express the uranium (or radium) required to cause the leaf to fall one division of the scale in a unit of time. The electroscope is standardized as follows:

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[a] Rutherford, E., and Boltwood, B. B., The relative proportion of radium and uranium in radioactive minerals: Am. Jour. Sci., ser. 4, vol. 22, 1906, p. 1.

[b] According to these figures 1 ton of 30 per cent ore would contain 102.7 mg. of radium calculated as chloride.

The emanation is separated and collected with the apparatus shown in figure 2. A small quantity of a standard sample of powdered uraninite or pitchblende of known uranium content is weighed out and put in the flask F, which has a capacity of about 50 c. c. Usually a quantity that contains 10 mg. of metallic uranium is convenient. A rubber stopper, fitted with a small dropping funnel and a short delivery tube, is then inserted in the flask and connections are made with the gas burette B, which has been previously filled with freshly boiled hot distilled water containing a little sodium hydroxide. Nitric acid (1:1) is then poured into the dropping funnel, the leveling reservoir of the gas burette is lowered below the level of the acid in the funnel, and then, by opening the pinchcock C and the stopcock of the funnel, most of the acid is allowed to flow into the flask. The stopcock is then closed, the leveling reservoir is replaced, and the flask is gently heated until the pitchblende is dissolved. By withdrawing the flame for a few moments, sufficient water is allowed to flow into the flask to continue the boiling for 10 minutes. The gas collected in the burette is then introduced into the electroscope which has previously been partly exhausted. After three hours, which is the time required for the radioactivity to attain a maximum, the rate of leak is determined.

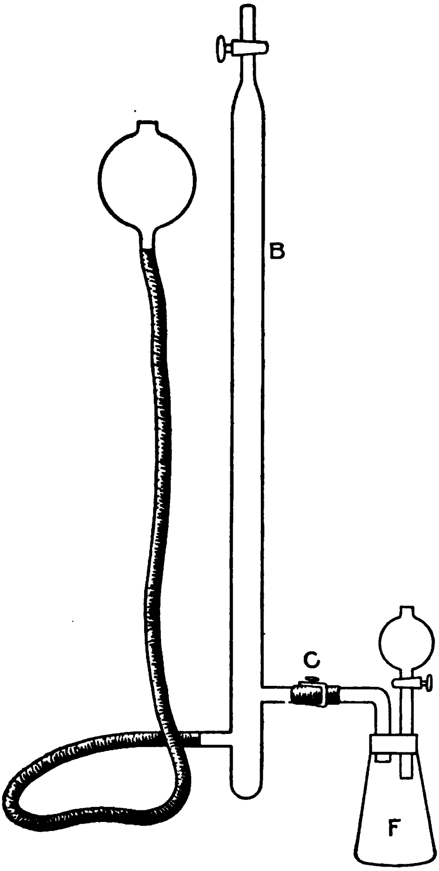

FIGURE 2.—Apparatus for separating emanation from uraninite.

CORRECTION FOR STANDARDIZATION.

A small correction, first indicated by Boltwood,[a] has to be made for each sample of pitchblende used for standardization, but 2 or 3 grams of the mineral, once this correction has been ascertained, will last for years. The correction is made as follows:

About 0.1 gram of the mineral is weighed and introduced into a small tube closed at each end by means of a piece of rubber tubing and a clip. The tube is allowed to stand until the emanation given out by the pitchblende is in equilibrium. One end of the tube is then connected with the previously exhausted electroscope and a current of air passed gently through the tube into the electroscope. This carries into the instrument the emanation that has been given out naturally in the cold by the pitchblende. The effect on the electroscope is noted and is compared with the effect from the emanation occluded by the pitchblende, the emanation being liberated by boiling with acid as described above. If one-tenth of the emanation is given out in the cold one-tenth more material must be used. For example, instead of taking a quantity of the pitchblende that contains 10 mg. of metallic uranium, a quantity must be used that contains 11 mg., but the calculations are made as though a quantity containing 10 mg. had been used.

METHOD FOR EXACT DETERMINATION OF RADIUM.

After the electroscope has been standardized and it is desired to determine the radium content of a mineral, a definite quantity of the mineral, depending on its uranium content, is fused with four or five times its weight of mixed sodium and potassium carbonates. In the case of a carnotite containing 2 to 4 per cent U_3O_8, about ½ gram of material should be used. The fused mass is lixiviated with water and washed with dilute sodium carbonate solution. The insoluble residue is dissolved in dilute hydrochloric acid. The alkaline and acid solutions are placed in separate flasks, which are then corked. Through the stopper of each flask passes a glass tube, which is closed by a piece of rubber tubing fitted with a small clip. On standing for one month, the radium emanation is re-formed by the radium in solution and is in equilibrium with the radium; that is, the maximum amount has been re-formed. The emanation in the two flasks is boiled off into the gas burette in a manner similar to that already described, but without the addition of acid, and the air containing the emanation is introduced into the electroscope. After 3 hours a reading is taken and from the data thus obtained the percentage of radium in the sample can be determined. An example of this method follows:

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[a] Boltwood, B. B., On the radioactivity of natural waters: Am. Jour. Sci., vol. 18, 1904, p. 381.

Sample determination of radium content of an ore.

A sample of pitchblende loses 10 per cent of its emanation at room temperatures. It contains 50 per cent metallic uranium. Therefore 22 mg. of the ore will, on dissolving in acid, liberate emanation in equilibrium with 10 mg. (0.01 gram) of metallic uranium. This emanation, 3 hours after introduction into the electroscope, causes the leaf to fall at the rate of 40.5 divisions per minute. The natural leak (0.5 divisions per minute) subtracted from this leaves 40 divisions per minute due to the emanation. Therefore the fall of one division per minute represents the total emanation associated with $\frac{.01}{40}=2.5\times10^{-4}$ grams of uranium in the mineral. This is the "constant" for the electroscope.

One gram of ore is fused with fusion mixture as already described. At the end of a month the emanation obtained from the two solutions is introduced into the electroscope. After 3 hours the rate of fall of the leaf is 18.5 divisions per minute. Subtracting the natural leak (0.5) leaves 18 divisions per minute. Therefore 1 gram of the ore contains $18\times2.5\times10^{-4}=45\times10^{-4}$ grams of uranium.[a] As one gram of uranium is in radioactive equilibrium with 3.4×10^{-7} grams of radium, 1 gram of the mineral will contain $(45\times10^{-4})\times(3.4\times10^{-7})=1.53\times10^{-9}$ grams of radium.

COMMERCIAL METHODS OF TREATMENT OF ORES.

Almost all the commercial methods which have been proposed for treating uranium ores, or ores carrying both uranium and vanadium, have been for the extraction of the uranium and vanadium only, without reference to the recovery of the radium. Some of these methods have been used in concentration only and not for obtaining refined products. In such cases the radium has usually been lost. The reason that no attempts have been made until recently to extract the radium, in addition to the other two metals, is probably because of operators not knowing the best methods to use, a lack of capital, or to a hesitation in entering a new and little-known field. The ores have been purchased abroad mainly for their radium content and the profit has been largely in connection with the extraction of the radium. This can be readily shown by considering gross returns on the refined products that would be obtained from working up a carnotite ore containing 3 per cent U_3O_8 and 4 per cent V_2O_5, which are fairly typical values for ore from the Paradox district.

Value of uranium and vanadium per ton of ore.[c]

3 per cent U_3O_8=60 pounds per ton; at $2.50 per pound	=$150
4 per cent V_2O_5=80 pounds per ton; at $0.75 per pound	= 60
60 pounds of U_3O_8=8.1 mg.[b] of radium chloride; at $90 per mg.	= 729
Total,	939

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[a] This is true only when the uranium and radium are in equilibrium. In other cases it represents the theoretical amount of uranium in equilibrium with the radium actually present. In pitchblende, since it is a primary mineral, the ratio of uranium to radium is constant, 1 gram uranium=3.4×10^{-7} grams of radium, or 2,000 pounds U_3O_8=342 mg. $RaCl_2$, and if the percentage of uranium is known by analysis, the amount of radium present can be calculated directly. In carnotite and other recent uranium minerals, the equilibrium ratio is not constant and the radium present has to be found by experiment.

[b] The theoretical content of radium chloride from 60 pounds of uranium oxide would be 9.2 mg.; 10 per cent has been allowed for the radium and uranium not being in equilibrium.

[c] The figures given do not take into consideration losses in treatment.

The refined uranium and vanadium products from a ton of the above ore would be worth $210, and the radium chloride extracted from a ton of the same ore would be worth $729, or more than three times the value of the uranium and vanadium. Therefore a method of extraction that would not be profitable if the uranium and vanadium alone were recovered might yield a profit if it recovered the radium also. It is difficult to give the exact cost of extracting and refining radium salts. This varies in different places, being governed by the cost of labor, chemicals, freight, and other items, but $20 per milligram is probably a maximum cost in Europe for extracting radium from carnotite ores. Assuming that the market value of radium remains at its present figure, namely, $90 per milligram for radium chloride, this leaves a wide margin of profit over the necessary cost of production in this country even though the cost be somewhat more than abroad.

The carnotite produced in the United States during the last two years has been the means of a sudden and large increase in the production of radium salts, and the market for radium at present is somewhat uncertain, but if prices should fall the margin is considerable. Any method, therefore, used in this country for the extraction of uranium and vanadium from carnotite ores, or of uranium from pitchblende, should include the saving of the radium in marketable form, even though it does not involve final purification of radium salts. The purification, however, of radium salts of medium activity, such as those containing from 10 to 50 per cent radium chloride, is not difficult. It is difficult to obtain chemically pure radium salts, but these in most cases are not at all necessary, the lower-grade material being just as valuable, if not more so, for most medical and other scientific purposes.

BLEEKER PROCESS FOR THE RECOVERY OF VANADIUM.

The method of treatment suggested by W. F. Bleeker[b] involves the production of copper, iron, or lead vanadate from vanadiferous ores and has for its object the expeditious and clean separation of the vanadium constituents of the ores. The process includes two general stages—the production of an approximately neutral vanadium solution, and the precipitation of this solution with the metallic substance. The first stage is performed in five successive steps: The ore is pulverized to a suitable degree of fineness; the pulverous matter is roasted after having been mixed with a flux to render the vanadium constituents soluble after roasting; the roasted product is leached with water to dissolve the alkali vanadate or vanadyl salts; the residue is leached with a dilute acid solution to dissolve any

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[a] United States patent 231,912.

remaining vanadium compounds; and the two solutions are mixed to obtain a neutral vanadium solution. The flux by which the vanadium constituents of the ore are rendered soluble after roasting is composed of sodium chloride and a fixed alkali, such as potassium hydroxide, sodium hydroxide, potassium carbonate, or sodium carbonate, preference being given to the potassium hydroxide. The product, after being roasted with this flux, is leached with water to dissolve the soluble alkali vanadates, and this rich alkaline liquor is led to storage tanks. The residue is again leached, but with a dilute mineral acid, such as hydrochloric, for the purpose of dissolving the vanadates insoluble in water, leaving a residue practically free from vanadium. The alkaline and acid liquors are mixed in such proportions as will produce a neutral solution. Copper sulphate or any other copper salt is added to the solution to make an insoluble precipitate of copper vanadate which is easily filtered.

This process was probably devised for the treatment of the "patronite" ores from Peru in the plant of the American Vanadium Co. These ores do not carry uranium. The process might, however, be used with carnotite ores and by a slight modification of it a large part of the radium recovered. A test of a small sample of carnotite ore made by the Bureau of Mines, in which two parts of ore were sintered with one part of sodium carbonate and one part of sodium chloride, showed that the total radium content present in the alkaline solution, the acid solution, and the insoluble residue was as follows:

Distribution of radium content of ore in Bleecker process.

	Per cent.
Alkaline solution	1.1
Acid solution	72.0
Residue	26.9
	100.0

The radium could be readily obtained from the acid solution by precipitating it with a moderate quantity of barium sulphate. This will precipitate the radium as radium sulphate, from which the radium can be recovered either by fusion with fusion mixture or by leaching with sodium carbonate, as described under methods of obtaining radium from pitchblende residues (p. 79). The loss indicated by the figures above is altogether too large for a commercial plant, but the yield might possibly be improved by using different proportions and by changing the conditions.

HAYNES-ENGLE PROCESS FOR THE RECOVERY OF URANIUM AND VANADIUM.

The process[a] of J. H. Haynes and W. D. Engle involves the treatment of ore containing either uranium or vanadium, or both of these metals. The ore is first crushed to twelve mesh, and is then boiled with a solution of alkaline carbonate, preferably sodium carbonate or potassium carbonate, until the uranium or vanadium, or both, in the ore is dissolved. The strength of the sodium-carbonate solution and the length of time necessary to boil are determined by the proportion of uranium and vanadium in the ore and will probably vary considerably. The originators of the process claim, however, that 100 pounds of sodium carbonate per ton of ore for each 1 per cent of uranium and vanadium, or either, present will give good results. Ordinarily the time required for boiling should be about one hour. After the uranium and vanadium, or either, are dissolved, the clear solution is drained into a separate tank. The uranium is precipitated as sodium uranate by the addition of sodium hydroxide to the solution. This precipitate is removed from the solution, which contains all of the vanadium. From this solution, either with or without neutralization, the vanadium is precipitated as calcium vanadate by the addition of water-slaked lime.

When the process was in actual operation with carnotite ores an extraction of 80 per cent of the uranium and 60 to 65 per cent of the vanadium was obtained.

The main object of this method is the chemical concentration of carnotite ores, involving the recovery of the uranium and vanadium. The radium, all of which passes into the insoluble residue, could be removed by the following treatment: After the residue from the alkaline solution has been thoroughly washed in order to remove all sulphates, that portion of the radium that has been converted into radium carbonate, which is insoluble in the alkaline solution, could be leached from the residue by dilute hydrochloric acid, and the radium chloride could be recovered from this solution without difficulty. If any of the radium still remained in the residue, it could be recovered by boiling a second time with the carbonate solution, washing as before, and leaching again with dilute acid. The commercial success of such a method would depend upon whether or not all of the radium could be leached out by the first treatment with acid, the proportion of sulphates and alkaline earths in the ore largely determining this point.[b]

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[a] United States patent 808,839.

[b] A patent has been recently taken out by W. F. Bleeker (U. S. patent 1,065,581), involving the recovery of the radium as indicated.

KOENIG PROCESS FOR THE RECOVERY OF VANADIUM.

G. A. Koenig's process[a] relates to improvements and processes for the recovery of vanadium from its ores particularly the vanadiferous sandstone of southwestern Colorado and other places. The method is based upon the fact that roscoelite, or ores of vanadium containing the same mineral substances, can be completely decomposed and dissolved by the action of a dilute solution of sulphuric, hydrochloric, or other acid under proper heat and pressure.

In practice, the originator uses a solution containing about 20 per cent sulphuric, hydrochloric, or other acid, at a temperature of about 200° C. and under a pressure of about 225 pounds per square inch, and claims that this process will completely decompose and dissolve the roscoelite within a few hours. The filtered solution is evaporated to a mushy consistence, placed in a retort or muffle and heated gradually to a bright red heat. This heating drives out the acid, which may be recovered in any suitable manner. The residue is a mixture of the oxides of vanadium, aluminum, manganese, iron, and other metals that may be present in the ore. When sulphuric acid is used sulphates of calcium, potassium, and other sulphates are also present. The mixture of oxides and sulphates is mixed with the proper quantity of sodium carbonate and is roasted at red heat in an oxidizing flame either with or without the addition of oxidizing agents. The roasted mass is disintegrated with boiling water and, while still in the boiler, is treated with carbon dioxide to precipitate the aluminum as aluminum hydroxide.

The acid methods and their value in the extraction of radium will be discussed later.

FISCHER PROCESS OF EXTRACTING VANADIUM FROM CARNOTITE.

In Siegfried Fischer's method of extracting vanadium from carnotite the ore is boiled with a solution of sodium or potassium hydroxide. The process comprises three distinct steps: The conversion of vanadium compounds insoluble in water into soluble form, leaching and separating vanadium from uranium, and precipitating and drying the vanadium compound. It is claimed that the dried product is ready for reduction to ferrovanadium and that the uranium residue is in marketable condition. By this process, working on crude ores, Fischer claims an extraction of 65 to 67 per cent of the vanadium present, and from concentrates carrying from 9 to 16 per cent vanadium an extraction of 93 to 94.6 per cent.

In this method the uranium would remain in the insoluble residue as sodium uranate with the radium, and the value of the sodium-hydroxide leach would be simply to extract the vanadium. An appreciable proportion of the sodium hydroxide would react with the silica present, involving a large waste of material. It does not seem to be a desirable method for the treatment of carnotite ores when the extraction of the radium is desired.

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[a] United States patent 986,180.

METHOD USED BY PRIMOS CHEMICAL CO.

The Primos Chemical Co., of Newmire, Colo., and Primos, Pa., roast roscoelite ore with common salt. The ore contains almost no uranium and no attempt is made to recover the minute quantity found in some of it. The ore is weighed and mixed with salt. The mixture is then coarsely ground and run into a drier which reduces the moisture content to 1 per cent. The material, which comes out somewhat caked, is ground to 20 mesh and roasted in a furnace for about three hours. The roasted material is delivered at the bottom of the furnace to conveyors, which carry it to lixiviation tanks, where it is treated with water and filtered. The vanadium is now in solution as sodium vanadate. A solution of ferrous sulphate is added to the filtrate, and the vanadium is precipitated as vanadate of iron. The precipitate is filtered out of the solution and dried. The dried material is sent to Pennsylvania, where it is reduced to ferrovanadium, probably by the Goldschmidt process. A complete extraction of vanadium is not obtained by this method.

If carnotite were treated in a similar manner the solution of the vanadium as sodium vanadate might be accomplished in the same way. The uranium would be converted into sodium uranate, which would remain in the residue. The radium would also be in the residue.

For this reason the method does not appear to be applicable to carnotite ores.

FLECK METHOD OF EXTRACTING URANIUM AND VANADIUM.

The American Rare Metals Co., of Denver, Colo., uses a method of extraction originated by Herman Fleck. The company's plant is in the McIntyre district south of Paradox Valley, and the crude sulphuric acid required in the process is hauled in wagons from Dolores. The finely crushed ore is treated with dilute sulphuric acid, which dissolves the uranium, vanadium, copper, and iron contained in the ore. The solution is decanted from the slime and sulphur dioxide gas is passed through it, reducing the iron and vanadium compounds to ferrous and vanadous forms. A calculated quantity of pulverized limestone (the rock is obtained a short distance from the plant) is then added until the metals begin to separate, calcium sulphate being precipitated. The solution is then decanted or filtered from this sulphate and the precipitation of the metals is completed by boiling with more pulverized limestone. A concentrate is thus obtained carrying about 20 per cent of uranium and vanadium calculated as oxides. Improvements in the process are contemplated by which a still higher concentration can be obtained.

The radium, of course, is in the undissolved slimes and the radium concentrate is obtained by fractionation of these slimes. A product carrying as high as 100 milligrams of radium per ton has been produced.

This process is in operation at the present time and recently a shipment of uranium, vanadium, and radium concentrates was made from Dolores.

RADCLIFF METHOD FOR COMPLEX RADIUM ORES.

Sidney Radcliff's method[a] relates to the economic commercial treatment of complex radium ores for the separate recovery as marketable products of the following substances: (*a*) Radium, as radium and barium sulphates; (*b*) uranium, as oxide or uranate; (*c*) the "acid earths," such as tantalum, niobium, titanium, as oxides; (*d*) the "rare earths," such as cerium, thorium, lanthanum, and didymium, as oxides.

The crushed ore or concentrates is fused with acid sodium sulphate. A powerful decomposing and oxidizing reagent is added to the fusion. The fused mass is cooled, pulverized, and thoroughly lixiviated and agitated with water, which dissolves certain constituents. The solution, which contains sulphates of radium, barium, and other elements in suspension, is led to settling tanks. The suspended and dissolved matter is treated as hereafter described for the recovery of the valuable constituents. In this manner the quantity of material to be precipitated with carbonate of soda is greatly reduced, thereby lessening the cost of treatment.

The various steps of the treatment are as follows:

The ore or the concentrate is crushed to pass a 30 to 40 mesh sieve.

The crushed ore is fused in a reverberatory or other suitable furnace with about 2½ times its weight of acid sodium sulphate. After the charge has been fused, and while the mass is still fluid, sodium chloride (10 or 15 per cent of the weight of the ore) is added and well rabbled. The addition of the sodium chloride and its reaction with or in the presence of the fused acid sodium sulphate causes a powerful decomposing and oxidizing effect and changes any ferrous sulphate to ferric sulphate.

The fused product is tapped from the furnace in the liquid state, cooled, crushed to powder, charged into suitable vats containing warm water, and agitated for some time. Most of the uranium, iron, and "rare earths," together with part of the titanium, niobium, and tantalum, go into solution. The radium is in the form of a sulphate which is insoluble and remains in suspension along with the sulphates of lime, lead, and barium, and the fine particles of gangue material.

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[a] United States patent 1,049,145.

The turbid liquid is rapidly siphoned into suitable settling vats. The coarse residues which remain in the bottom of the vat are washed several times with warm water and then rejected, the washings being passed into the settling vats.

From the settling vats two products—(*a*) solution and (*b*) fine slime—are obtained.

The solution (*a*) contains in the case of certain ores iron, aluminum, chromium, and uranium compounds, as well as compounds of the "acid earths" and "rare earths." Sodium carbonate is added, but not in excess, to the solution and all the above-mentioned elements are precipitated. The precipitate is removed by means of a vacuum filter and the clear liquid rejected. The precipitate recovered on the filter is boiled with an excess of a solution of sodium carbonate, which causes the uranium to pass into solution. The solution is filtered off and the uranium is then recovered as sodium uranate by the addition of sulphuric acid or caustic soda. The balance of the precipitate, which still contains, in addition to compounds of iron and aluminum, "acid earths" and "rare earths," is treated with moderately dilute sulphuric acid, which dissolves all of the precipitate with the exception of the "acid earths." The latter are filtered off, washed, and ignited. The washings are added to the sulphuric acid filtrate. This filtrate is treated with oxalic acid, which precipitates the "rare earths" as oxalates. These oxalates are then washed, dried, and ignited.

The fine slimes (*b*) contain most of the radium and are treated according to the usual methods. The crude radium and barium sulphates are obtained in the ordinary way.

The objections to this method, in the treatment of low-grade carnotite ores, are the inconvenience and cost of fusing an ore that contains as much as 95 per cent of silica, iron, and calcium compounds in order to obtain the average 5 per cent of uranium and vanadium compounds present.

TREATMENT OF CARNOTITE ORES WITH NITRIC OR HYDROCHLORIC ACID.

The simplest way to extract uranium, vanadium, and radium from a carnotite ore is to treat the ore or concentrate directly with boiling concentrated nitric or muriatic acid. Even the vanadium and silica combinations can be decomposed by boiling with these acids (1:1) for an hour, nitric acid giving the better results. Carnotite itself is soluble in cold dilute hydrochloric or nitric acid. The practicability of such a process depends on its cost and the possibility of readily recovering the radium. The first step is what counts in the main in treating these ores. After that, the separation of the uranium and vanadium is a simple matter, several methods being available. Nitric acid would be absolutely out of the question for use in a commercial process if the vanadium and uranium alone were recovered, but the large additional cost of nitric acid over sulphuric and hydrochloric acids might be more than offset by the saving of labor and time if by its use a convenient and easy method of extracting the radium in addition to the uranium and vanadium could be devised. The price of hydrochloric acid being very little more than that of sulphuric, a small additional saving in the manufacturing process will justify its use instead of the latter acid.

Some tests made on a small scale on a typical sample of carnotite ore from Paradox Valley, containing 2 per cent U_3O_8 and about 2½ per cent V_2O_5 showed the following extraction of radium.

Extraction of radium from a carnotite ore.

With hot concentrated commercial muriatic acid, 97.1 per cent of the radium in the ore went into solution and 2.9 per cent remained in the residue.

With hot dilute (1 : 2) muriatic acid 88.4 per cent was dissolved in the acid and 11.6 per cent remained in the residue.

With hot, concentrated, commercial nitric acid, 97.5 per cent of the radium went into solution and 2.5 per cent remained in the residue.

With hot dilute (1 : 2) nitric acid, 96.8 per cent of the radium was dissolved and 3.2 per cent was left in the residue.

The concentrated acids in both cases gave the better extraction, although the dilute nitric acid was not much inferior to the concentrated. The radium can be easily recovered by further dilution and by the precipitation of barium sulphate in the solution. The radium which is carried down with the barium sulphate as radium sulphate can be recovered by leaching the precipitate with a solution of sodium carbonate, or by reducing the mixed sulphates to sulphides by heating the precipitate in a furnace in a current of coal gas, then dissolving the sulphides in hydrochloric acid and fractionating. Although a better extraction is obtained with nitric acid, its use in addition to the increased cost has some disadvantages. Nitric acid is unpleasant and difficult to handle, although porcelain-lined ware is now made which withstands the action of hot nitric acid exceedingly well. Also a precipitate of barium sulphate does not carry down the radium as well from dilute nitric acid solutions as from dilute hydrochloric acid solutions. The figures given above were obtained with the same sample of ore, which contained only a small proportion of sulphate, although the acids themselves contained some sulphate. In the case of an ore with a large gypsum content, the extraction, especially with hydrochloric acid, would be much lower than it was in this case.

The same acid could be used in treating successive portions of ore until the strength became reduced to a certain point. In addition to their uranium and vanadium content, all of these ores carry a certain proportion of iron and calcium, as well as traces of other metals. The total acid-soluble material in the ore treated as mentioned above was about 12 per cent. For some ores this figure would be higher and for others lower. The results of further investigations in this direction are to be published by the Bureau of Mines.

Hot concentrated nitric acid can be used for dissolving pitchblende and changing both the uranium and radium into soluble form, provided the ore does not carry too much iron or copper pyrite. After separation and decantation from the insoluble residue, the acid can be diluted and the radium recovered with barium sulphate by precipitation. If this method is not desirable, the radium can be obtained mixed with salts of barium, strontium, or calcium, according to the method of separation that is chosen.

BLEEKER PROCESS OF SEPARATING URANIUM FROM VANADIUM.

A process[a] is described by W. F. Bleeker for the separation of vanadium from uranium. It is stated that this process is applicable for treating any solution of uranium and vanadium containing an alkaline carbonate, such as sodium carbonate, ammonium carbonate, or potassium carbonate. The solution is first heated to a temperature preferably not exceeding 90° C., and heated sodium hydroxide is added in sufficient quantity to precipitate uranium as a mixture of sodium uranate and uranyl hydrate. This precipitate carries vanadium and the resultant solution also carries some uranium which may be recovered by any suitable subsequent treatment. The precipitate is filtered and washed with water, dissolved in acid, preferably sulphuric, forming a solution of uranyl sulphate. This solution is treated with an excess of sodium carbonate so as to make it slightly alkaline. It is then electrolyzed in a tank, the anodes being of any desired active metal, such as iron, copper, or nickel, and the vanadium is precipitated as the vanadate of the anode metal. For example, if a nickel anode is used, the product would be vanadate of nickel. The uranium remains in solution. After the solution has been electrolyzed long enough to precipitate all of the vanadium, it is filtered, and the uranium salts, free from vanadium, may be precipitated by any well-known process.

PROCESS OF EXTRACTING VANADIC ACID FROM COPPER VANADATE.

Bleeker also describes a process[b] of extracting vanadic acid from insoluble copper vanadate. Copper vanadate is decomposed by a dilute mineral acid, preferably sulphuric, the product obtained being an acid solution of copper and vanadium. The insoluble vanadic acid (V_2O_5), which is about 80 to 90 per cent pure, is separated by filtration, leaving copper and vanadic acid in solution ($Cu+V_2O_5+H_2SO_4$). The copper is extracted from this solution, and the vanadic acid obtained by electrolysis. The cathode is of copper, and the anode is preferably carbon or platinum. The copper is deposited on the cathode, and the vanadium remains in solution ($V_2O_5+H_2SO_4$). By evaporating all or part of the sulphuric acid from this solution, the vanadic acid may be recovered. The mother liquor upon dilution is still strong enough to be used in the process of producing copper vanadate.

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[a] United States patent 1,050,796.

[b] United States patent 1,049,330.

AUSTRIAN METHOD OF EXTRACTING RADIUM FROM PITCHBLENDE.

In the method used for treating pitchblende in Austria, the pitchblende is fused with sodium sulphate, and the uranium is thus changed to sodium uranate, which can be dissolved by means of dilute sulphuric acid. The residue contains all of the radium. Before the discovery of radium this residue was considered to be a waste product and was thrown away. The extraction of the radium from the accumulated residue is well described by Haitinger and Ulrich[a] and is, with some minor changes, similar to the method now used by the Austrian Government. The work was done in the laboratory of the Austrian Incandescent Gaslight and Electric Co., and 10,000 kilograms (22,000 pounds) of pitchblende residues were treated, which represented about 30,000 kilograms (66,000 pounds) of pitchblende containing 53.4 per cent of U_3O_8. Moisture in the various shipments varied from 10.3 to 18.4 per cent. This moisture was driven off at a temperature of 105° C., the 10,000 kilograms of residue losing 1,340 kilograms in this way. The work took two years, due to time spent on analyses at the start and to experiments to obtain the best results. The method given was developed as being most suitable to the equipment available. Five thousand kilograms annually could be treated.

The chemical operation involved, first, the decomposition of the residues; second, the removal of the radium sulphate by precipitation; and, third, the solution and concentration of the latter. The first step consisted of digesting the residue with sodium hydroxide, 100 kilograms being boiled for one day with a solution of 50 kilograms of hydroxide in 200 liters of water. Forty per cent of the alkali was converted to sulphate and to silicate. The solution contained some radium, but the total radium so dissolved from the entire 10,000 kilograms of residue represented the radium equivalent of only 10 kilograms. Therefore this solution was thrown away.

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[a] Haitinger, Ludwig, and Ulrich, Karl, Bericht über die Bearbeitung der Pechblende Rückstände: K. K. Akad. Wissenschaft., 1908, vol. 117, p. 619.

After the boiling the residue was allowed to settle and the liquid was decanted. The residue was washed to remove the greater part of the sulphates, filtering and washing being done in a funnel with a capacity of 100 kilograms, which had a lead suction tube 3 meters long. The vessel containing the washed residue was placed on a water bath and the residue was treated with an equal weight of dilute (1:1) crude hydrochloric acid. After prolonged heating the acid solution was decanted and the residue washed with water. This water was then used to dilute the next portion of acid for a new sample.

Crystals of calcium sulphate and lead chloride formed in the acid solution as it cooled. Neither the solution nor the crystals contained an appreciable quantity of radium, but nearly all the polonium and actinium was in the solution. It was therefore treated with ammonium hydroxide to precipitate the polonium and actinium. The filtrate was not radioactive and was discarded.

The residue from the treatment with crude hydrochloric acid was boiled in a solution of sodium carbonate, the carbonate (made by the ammonia process) being free from sulphates. Fifty kilograms of sodium carbonate in 200 liters of water were used for a 100 kilogram sample. By this treatment a large part of the radium sulphate was converted to radium carbonate. Therefore, in subsequent treatments the solutions had to be kept free from sulphate. The residue was washed till free from all trace of sulphate and then treated with pure hydrochloric acid. The boiling with soda and the treatment with acid was repeated three times. After the third treatment only 2 per cent of the original radium content remained in the residue which was thrown away. The soda extracts were practically free from radium. The washing of each of the various residues consumed from 4 to 6 weeks. The hydrochloric acid extracts, containing nearly all the radium, were united and the radium was precipitated as a sulphate with sulphuric acid. Besides the radium the precipitate contained, of course, the alkali earths, including calcium, much lead containing radio-lead, and a small quantity of rare earths containing actinium. The sulphate precipitate, called crude sulphate, represented from 0.5 to 2 per cent of the weight of the original residue taken.

The crude sulphate was reconverted to carbonate by repeated boiling with sodium carbonate solution. All of the sulphate from any one sample could not be converted and the final residue from the treatments was, therefore, not thrown away, but added to a new portion of crude sulphate. After each carbonate treatment, an extraction with hydrochloric acid was made. The lead chloride formed in the solutions was removed and freed from radium by repeated crystallization in hot water. Sixty kilograms of lead chloride were thus obtained from the entire 10,000 kilograms of residue. It was saved and treated for its content of radio-lead.

The hydrochloric acid solutions from the crude sulphate were freed completely from lead by hydrogen sulphide and were then evaporated to dryness on a steam bath. The calcium chloride in the residue so obtained was dissolved in concentrated hydrochloric acid, in which barium chloride is only slightly soluble and radium chloride is still less soluble. The residue remaining, called crude chloride, consisted of radium and barium chlorides, with some strontium and calcium chlorides and traces of other impurities.

From this point on the concentration was continued by fractional crystallizations from water solutions. Radium chloride, which is the least soluble of the chlorides, accumulated in the crystals, the foreign matter remaining more and more in the mother liquor. The first fraction was, of course, the richest in radium. Two steps had to be watched in this process—first, the separation from the system of as large a quantity of radium-free barium chloride as was possible; second, the making of a relatively large first fraction. This second step can be taken by temporarily stopping the crystallization of the first series until the crystals of the second series are of sufficient radioactivity to be united with the first.

The crystallizations were all carried out on a steam bath in order to avoid contamination with sulphate, as might have been the case if heating had been done with a direct flame.

Finally, two portions of crystals were obtained, one of about 2 kilograms, containing nearly all of the radium, and the other of about 11 kilograms, with very little radium.

The 2-kilogram portion was treated as raw material for the production of radium chloride free from barium. After this had been crystallized about thirty times, the first fraction of about 9 grams was further crystallized, and the lower fractions were combined into three groups according to their activity. The 9-gram portion was first purified with hydrogen sulphide, removing traces of lead which probably came from the glassware. Further work was conducted in quartz vessels. The salt was dissolved in dilute hydrochloric acid, warmed and allowed to crystallize. Four fractions were so obtained.

Atomic weight determinations were made with three of the fractions, the values obtained being 143.2, 185.2, and 225. The latter represented practically pure radium chloride.

Some of the lower fractions of barium chloride that were poor in radium were converted to the bromide and then fractionated. Only one portion, that which should contain the most radium—that is, the last of the four analagous fractions—was saved. The other portions were reconverted to chloride and added to the main chloride crystallization system. In all, 3 grams of pure dry radium chloride and 0.236 gram of radium bromide were obtained from 10,000 kilograms of original residue.

ANALYTICAL METHODS FOR URANIUM AND VANADIUM.

Operators in the Colorado and Utah uranium and vanadium fields had considerable difficulty during the past year in having correct analyses made. As a rule, chemists have been able to check their figures on vanadium, but uranium assays from different men have frequently shown widely divergent results. A variation of over 100 per cent has not been uncommon in some cases. As it has been extremely difficult, if not impossible, to sell ore containing less than 2 per cent $U_8 O_8$, a small error on an ore of this character may mean the difference between selling and not selling. Since the results obtained have been so variable, the foreign buyers are suspicious of such results unless the analyses were made by two or three firms in whom they have confidence. The difficulty has been a tendency to use methods that may work fairly well with some uranium ores, but are of too general a character to be adapted to the large variety of ores associated with carnotite.

METHODS FOR THE ANALYSIS OF CARNOTITE.

The Bureau of Mines has not as yet tested out the analytical methods which are at present in use. Dr. Hillebrand of the Bureau of Standards, whom the writers asked to suggest the best methods of analyzing carnotite, has kindly made the following suggestions:

METHOD FOR THE DETERMINATION OF URANIUM AND VANADIUM.

In the first place, a knowledge of the mineral composition of these ores and of the behavior of their valuable mineral components toward solvents is important. There are several vanadium ores in Colorado and Utah besides pitchblende and the carnotite ores. For instance, there are yellow and green vanadates of copper, barium, and calcium; red vanadates of calcium, sometimes taken for the vanadic acid; and a black ore, rich in vanadium, which carries that element in three states of oxidation. Of all these I have samples and hope to describe them all. The carnotite minerals themselves are perhaps in places accompanied by one or other of the above. Always associated with carnotite, so far as my experience goes, are potassium-vanadium-aluminum silicates, sometimes the green roscoelite and again a gray amorphous powder mentioned by me in my paper on carnotite.[a] Both of these hold vanadium as V_2O_3. These silicates are but slowly decomposed by mineral acids, whereas the carnotites dissolve with great ease in even very dilute acids.

This difference of behavior makes it possible to simplify the analysis of such ores, so far at least as the uranium is concerned, for a few minutes' treatment with cold and dilute acid (best nitric) extracts all the uranium almost free from iron and aluminum and accompanied by that vanadium only with which it was combined in carnotite.

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[a] Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, pp. 30, 31; Am. Jour. Sci., 4th series, vol. 10, 1900, p. 144.

The remainder of the vanadium in silicate combination can be extracted by boiling for half an hour or so with nitric acid (say, 1:1). The silica of the silicates separates for the most part instead of dissolving in the acid, and in a form that is not gelatinous. It is thus easy to get both the valuable constituents of the ore into solution free from the great mass of quartz and silicate silica without resort to an alkali fusion. In this the vanadium can be determined in any convenient way.

If one prefers, the ore may be at once attacked by the hot acid.

If the cold acid treatment is used my practice is as follows, wherein it must be understood that I aim at the composition of the carnotite and not at the total vanadium in the ore: Evaporate the entire solution to dryness, take up with cold water and filter through a small filter, wash with a little cold water but without transferring the residue to the filter. The filtrate holds almost all the uranium and all the strong bases as nitrates, the vanadium remaining undissolved. The evaporation to dryness with nitric acid may be repeated on the filtrate. If the dry deposit shows a red color, repeat the extraction with water.

Dissolve the residues in a very little nitric acid, transfer the solution by degrees to a capacious porcelain boat, in which it is carried to dryness on the hot plate. Put the boat in a glass tube provided with two large U tubes with water enough in them to seal the bend. Pass HCl gas (from HCl solution and H_2SO_4) through the tube. Instantly brown red fumes of an oxychloride of vanadium come off in great volume. These are held quantitatively by the U tubes whose liquid contents by and by become saturated with HCl.

When the escape of brown fumes ceases, draw the boat out, add nitric acid, evaporate again to dryness, and treat anew with HCl gas, using U tubes with fresh water. Repeat these operations if need be till there is no further evidence of vanadium in the boat. It may be advisable to heat the tube somewhat toward the end of the second or third distillation.

Evaporate the combined distillates with H_2SO_4 to expel the HCl, dilute, introduce H_2S into the blue liquid to complete the reduction of the V_2O_5 and to precipitate Mo and As if present, filter, expel the H_2S, titrate with $KMnO_4$, reduce with SO_2 gas (not solution), boil, and titrate again as a check. The SO_2 value is usually a little lower and is the one to be accepted.

Convert the contents of the boat to nitrates and add the solution to the main one. Add H_2S water drop by drop to throw out Cu (or Pb), then a drop or two of H_2SO_4 to precipitate barium; after a time, filter. Uranium can then be determined gravimetrically after separating from alkalies, lime, iron, and aluminum, which separations offer no difficulties if properly carried out. The little phosphorus present is with the ignited U_3O_8 (partly at least) and can be determined and deducted.

These last operations can be shortened, if no arsenic is present, by separating iron and aluminum at once by ammonium carbonate and ammonium sulphide, evaporating the filtrate first with HNO_3, then with H_2SO_4, to expel HNO_3, reducing the sulphate solution with zinc and titrating with $KMnO_4$.

All this seems perhaps more complicated than it is in reality, and the operations are not very long, even when repeated. It should be borne in mind, too, that with such valuable ores as these it pays to expend some extra time to insure good results. Moreover, no solid reagents are introduced at any stage of the analysis.

There are some things to be learned, of course, about the distilling operation. The presence of much iron or aluminum retards very much the expulsion of the vanadium, so the method is not to be recommended when these are present in some amount. Care must be taken that there is not enough liquid in the U tube nearest the boat to allow of its being explosively sucked back if the gas current slackens too much before the water becomes saturated. A rapid gas current is desirable. The red fumes liquefy in part in the tube, but if not finally driven over can be washed out with water after withdrawing the boat.

This method will give exact results if properly carried out, but it is probably too much to expect of the average assayer who has to do things in a hurry.

Hillebrand also suggests the following methods of analysis, which are applicable to the carnotite ores, but not to pitchblende. He states that he has not tested them in all their details, and the methods may be varied somewhat to suit the needs of a particular ore when the analyst has intelligent knowledge of the mineral composition and has sound judgment.

METHOD FOR THE DETERMINATION OF VANADIUM.

Roast the ore gently if it carries organic matter, boil for 15 minutes with HCl, filter; ignite insoluble residue, add a few drops H_2SO_4 and evaporate several times with HF. Heat till fumes of H_2SO_4 come off, add more H_2SO_4, heat, and bring all into solution with H_2O, or fuse the residue with Na_2CO_3 and leach with H_2O and add filtrate to the first one. Evaporate with H_2SO_4 till fumes escape, take up with H_2O and precipitate with H_2S (hot at first, cold later); filter and wash; expel H_2S, oxidize with H_2O_2, evaporate to fumes of H_2SO_4, then several times with strong HCl to reduce V_2O_5 to V_2O_4, take up with H_2O, and titrate with $KMnO_4$ at 60° to 70° C.

The original directions of Campagne (as applied to steels) are to reduce with HCl before fuming with H_2SO_4; otherwise reduction of the V_2O_5 is not quite complete, but experiments by other chemists in this country seem to show that in presence of iron the reduction of V_2O_5 to V_2O_4 in these ores is complete when the order is reversed. Care must be taken, however, in the final fuming with H_2SO_4 not to prolong this beyond the time required to expel the HCl, for there is a slow oxidation of the V_2O_4 after that point is reached.[a]

METHOD FOR THE DETERMINATION OF URANIUM.

Treat the ore with cold HCl[b] (about 1:1) for 15 minutes, filter, precipitate with H_2S (in heat, and for a long time if arsenic is present), filter, expel H_2S, evaporate to dryness and convert to nitrates by evaporating to dryness with HNO_3. Treat the residue with cold water and filter at once through a small filter, washing with a little water. This takes most of the uranium nitrate into solution, but little or no vanadium.

Redissolve the residue in nitric acid and evaporate to dryness in a small porcelain dish. Pour NH_4OH over the residue and let stand some time to dissolve most of the V_2O_5; filter through a small filter. Redissolve the residue in HNO_3, evaporate again to dryness, and extract with NH_4OH a second time. Dissolve the final residue in HNO_3 and add to the solution of uranium nitrate. Neutralize this with NH_4OH and add $(NH_4)_2CO_3$ and NH_4HS.[c] Allow to stand 24 hours in a small stoppered flask, filter, wash with water containing a little $(NH_4)_2CO_3$ and NH_4HS. Evaporate the filtrate in a large porcelain or platinum dish, ignite gently, redissolve in HNO_3, filter if necessary, and then precipitate the uranium with ammonia that is free from carbonate. Filter and wash two or three times with water containing a little carbonate-free ammonia. Redissolve, reprecipitate, and wash as before. Ignite and weigh as crude U_3O_8. Warm with very little HNO_3 till the U_3O_8 is decomposed, filter from the insoluble residue (it may contain a little silica, iron oxide, and alumina), and deduct the weight of this from that of the crude U_3O_8.

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[a] If it is known from the character of the ore that the treatment with HCl extracts all the vanadium, the insoluble matter needs no further treatment.

[b] The recommendation of HCl, instead of HNO_3, is based on the possibility of arsenic being present, and of the need of precipitating it from a hot solution by H_2S.

[c] The ammonia extraction of the early nitrate residues might perhaps be omitted, but I recommend it strongly because we thereby reduce to a very small amount the V_2O_5 that is finally weighed with the U_3O_8, and also avoid other possible interferences. The ammoniacal extracts of the V_2O_5 contain, according to my own tests, mere traces of uranium.

Divide the filtrate into two portions. Test one for P_2O_5, and if that is found deduct its amount from the crude U_3O_8. Evaporate the other portion with H_2SO_4, take up with water, and filter from $BaSO_4$, if present. If $BaSO_4$ is present, deduct its equivalent of BaO from the crude U_3O_8. Next reduce the V_2O_5 accompanying the uranium by a current of SO_2 gas (not solution of SO_2) or by pure ammonium bisulphite, boil out SO_2 in a current of CO_2, and titrate at 60° to 70° C. with $KMnO_4$. Deduct V_2O_5 so found from crude U_3O_8 and call the remainder U_3O_8. Or, the second part of the nitrate solution may be evaporated with HCl to reduce V_2O_5 to V_2O_4, then fumed with H_2SO_4, filtered to remove $BaSO_4$ if present, and titrated for V_2O_4 as in the method for vanadium.

The fact that in the crude U_3O_8 the uranium may not be wholly U_3O_8, but in small part UO_3, does not invalidate the method for commercial needs.

Ledoux & Co., of New York, who have had much experience in the analysis of these ores, have furnished the details of the methods they use. They are as follows:

VOLUMETRIC METHOD FOR URANIUM IN CARNOTITE AND VANADIFEROUS URANIUM ORES.

The method given below is substantially Engle's method,[a] but some changes in manipulation have been introduced which render it more accurate.

The method[b] depends upon the separation of uranium as a phosphate from iron, vanadium, and other metals, the susceptibility of uranium phosphate to reduction by zinc and reoxidation by permanganate in a cold acid solution, and titration of the reduced uranium in sulphuric-acid solution by standard permanganate.

Many chemists who have experimented with the zinc-reduction method have laid stress on the difficulty of completely reducing uranium solutions and the danger of reoxidation by atmospheric oxygen during titration. It is our experience that reduction from UO_3 to UO_2 is easily attained, and further, that very prolonged reduction is apt to lead to formation of a lower oxide (UO ?). This lower oxide is oxidized with great rapidity by atmospheric oxygen to the uranous condition, but solutions of uranous sulphate are stable in the presence of air and may be exposed to it with impunity. In fact, they may be agitated with air for several minutes without altering the state of oxidation in a measurable degree. The method is conducted as follows:

Treat 5 grams of the ore, or a quantity containing not over 0.3 gram of metallic uranium, in a No. 3 beaker with 10 c. c. of HNO_3 (1.42) and 20 c. c. of H_2SO_4 (1.84). Cover with a watch glass supported above the beaker with glass hooks and evaporate till white fumes of H_2SO_4 begin to come off. Wash the cover with a little water, remove it, add a little more water to the beaker, mix the contents well and evaporate them almost to dryness, leaving a slightly moist residue containing about 2 c. c. of free H_2SO_4. This operation is easily conducted on a good hot plate; it does not require much time if the temperature is sufficient and the draft is good.

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[a] Engle, W. D., Western Chem. and Met., Nov., 1908.

[b] Pullman, O. S., jr., The determination of uranium and uranyl phosphate by the zinc reductor: Am. Jour. Sci., 4th series, vol. 16, 1903, pp. 229-239; and Kern, E. F., The quantitative separation and determination of uranium, Jour. Am. Chem. Soc., vol. 23, 1901, pp. 710-718.

The object of using a large excess of nitric and sulphuric acids is to destroy organic matter, which is present in many samples. The purpose of adding water after the first evaporation to fumes is to break up nitroso-sulphuric compounds and to completely eliminate nitric acid.

Add 75 to 80 c. c. of water to the residue and heat until all soluble matters are dissolved; then, without filtering, pass H_2S gas into the warm solution until all members of the H_2S group are precipitated. Filter and wash with warm water, collecting the filtrate and washings, which may measure 150 c. c., in a No. 3 beaker. Boil until all excess of H_2S is expelled and add H_2O_2 until all iron is oxidized. Neutralize the liquid with Na_2CO_3 and add about 2 grams in excess, then add 10 c. c. more of H_2O_2 and boil for 15 minutes. Filter through a 12.5-cm. S. and S. No. 489 paper and wash the precipitate four or five times with hot water, stirring it well with the jet of the wash bottle. Collect the filtrate and washings in a No. 5 beaker. Wash the precipitate from the filter into the No. 3 beaker with a fine stream of water, neglecting the small part that may adhere pertinaciously to the paper. Dissolve the precipitate in a little dilute H_2SO_4, neutralize as before with Na_2CO_3, adding about 2 grams in excess, add 10 c. c. of H_2O_2 and boil again as before. Filter through the same paper into the No. 5 beaker and wash the precipitate three times with hot water.

The combined filtrates and washings from the two Na_2CO_3 precipitations will contain all of the uranium and most of the vanadium. Add 5 grams of ammonium phosphate and 10 c. c. of H_2SO_4 to the liquid, boil it well until all CO_2 is expelled, make it slightly alkaline with ammonia and then slightly acid with acetic acid. Uranium is precipitated as ammonium-uranium phosphate and vanadium is retained in solution. Cool the beaker for three-quarters of an hour in ice water or let it stand over night. This precaution is necessary for the complete separation of the uranium precipitate. The precipitation appears to be more complete in faintly acid solutions than it is in ammoniacal liquids; hence the addition of acetic acid.

The precipitate is slimy and difficult to wash. Filter through a 12.5-cm. No. 589 S. and S. paper which has been treated with an emulsion of paper pulp. The paper pulp aids filtration and prevents the precipitate from passing through the paper.

Let the precipitate drain on the filter, and then from a wash bottle direct a stream of a dilute solution of ammonium sulphate (2 per cent) toward the inside seam of the filter paper at the top of the precipitate. The precipitate can thus be made to open affording a better drainage. Rinse the beaker with ammonium-sulphate solution and wash down the filter from the top until the paper is full. The precipitate is rather impervious and the object of this washing is to remove vanadium-bearing solution from the beaker and the filter paper rather than actually to wash the precipitate. After the filter has drained, remove as much as possible of the precipitate from it with a glass rod, transferring it to the beaker. Wash the filter well from the top to remove vanadium from it. While the filter is draining, add a little ammonium-sulphate solution to the beaker and beat the precipitate to a thin paste with it, breaking up all aggregations; then add about 100 c. c. of ammonium-sulphate wash-solution, stir well and filter as before. This method of washing is fairly rapid, and for large precipitates is more effective than washing on the filter would be; small precipitates may be washed on the filter.

Transfer the precipitate with a glass rod as completely as possible to the beaker and dissolve what remains on the paper in a little hot dilute (1:4) sulphuric acid, letting the acid run through the paper into the beaker containing the precipitate. Wash the paper well with hot water so that it may be used for filtering the second precipitation. See that all of the uranium phosphate in the beaker is dissolved in the acid. A little paper pulp will, of course, remain. Add 2 grams of ammonium phosphate, dilute to 250 c. c. with warm water, make slightly alkaline with ammonia, boil cautiously for several minutes and then make slightly acid with acetic acid. Cool the liquid as before and filter through the same filter, washing the beaker and the filter paper several times with ammonium sulphate solution. This procedure removes all or all but a small trace of vanadium in the case of the usual grade of uranium-vanadium ores, such as contain not over 4 per cent U_3O_8 and less than 6 per cent V_2O_5. In the analysis of richer ores a third precipitation as phosphate may be required to remove every trace of vanadium.

Dissolve the uranium phosphate through the filter paper in hot sulphuric acid, using in all not over 40 c. c. of dilute (1:4) acid. Wash the filter well with water, let the solution cool and test it with a few drops of H_2O_2 to make sure that no vanadium is present. Since the solution is dissolved through the filter, the liquid should be free from any suspended paper pulp. Add an excess of a strong solution of $KMnO_4$ and heat the solution to boiling. The object of adding $KMnO_4$ is to destroy sugars and other organic compounds formed by the action of the solution on cellulose and traces of starch from the filter papers.

Omission of this precaution will often cause variations in the titration of solutions that have been passed repeatedly through filter paper.

Cool the liquid, dilute it to a volume of 150 c. c. and pass it through a reductor in the same way as in determining iron. The reductor we employ is simple.[a] The zinc column is 19 cm. long and 2 cm. in diameter, of 30-mesh granulated zinc that has been amalgamated by treatment with a dilute solution of mercuric chloride. The reductor is worked by suction from a filter pump, and the precautions usual in making iron determinations are observed. The time required to pass the solution through the reductor is about 2 minutes, exclusive of washings.

After reduction, shake the solution vigorously for 1 minute with free access of air and titrate with a standard solution of $KMnO_4$. Subtract 0.5 c. c. from the burette reading for an $\frac{N}{20}$ solution of permanganate as a correction for end point and reductor errors.

GRAVIMETRIC METHOD FOR VANADIUM AND URANIUM IN CARNOTITE AND OTHER ORES.

Treat from 2 to 5 grams of ore, according to the proportion of vanadium, iron, and uranium present, in a covered beaker, with 10 c. c. of HCl and let it stand 15 minutes, shaking it occasionally. Add 5 c. c. of HNO_3 and heat on a steam bath. When the solution is quiet, remove the cover and evaporate to dryness. Add 3 c. c. of HCl and 5 c. c. of water to the residue and let it stand on the steam bath for a few minutes, stirring occasionally. Dilute with 25 c. c. of hot water, filter into a small beaker, and wash the residue with warm water.

Some ores do not yield all the vanadium to this treatment, a little remaining with the insoluble residue. To make sure that all vanadium is in solution, ignite the residue in a platinum dish, treat it with 5 c. c. of HF, and evaporate to dryness on a steam bath. Do not bake the residue, for it is not necessary to expel all SiO_2. Add 3 c. c. of HCl to the residue from the HF treatment and evaporate to dryness. Repeat this treatment to insure expulsion of HF. Treat the residue with 2 c. c. of HCl and 2 c. c. of water and stir with a glass rod until any red crust is dissolved, then dilute the solution with water and filter it into the main liquid.

Pass H_2S into the liquid to separate copper, lead, and other metals of this group, filter and boil the liquid to expel the H_2S. Concentrate the liquid to 100 c. c. if necessary, oxidize it with an excess of H_2O_2, and then neutralize with dry Na_2CO_3, adding 2 or 3 grams in excess. Boil the liquid for about 15 minutes until the yellowish uranium precipitate dissolves, leaving a brown precipitate which is principally iron. Filter and wash the iron precipitate with water, reserving the filtrate. Dissolve the iron precipitate in the least possible amount of HNO_3 (1:1), and add 10 c. c. of H_2O_2, neutralize with Na_2CO_3, add an excess of 2 grams of Na_2CO_3, and boil as before. Filter into the beaker containing the first filtrate. The iron precipitate may contain a little vanadium—reserve it for further treatment.

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[a] Blair, Analysis of iron and steel, 7th ed., p. 94.

Evaporate the united filtrates from the iron precipitation to a volume of about 200 c. c., add 10 c. c. of strong HNO_3 and boil until all CO_2 is expelled. Neutralize the free acid with ammonia (until a slight permanent precipitate appears), then add 4 c. c. of HNO_3 for each 100 c. c. of liquid. Now add 10 c. c. of a 20 per cent lead acetate solution, and enough of a strong solution of ammonium acetate to neutralize the nitric acid present and substitute acetic acid for it. The object is to precipitate the vanadium as lead vanadate in an acetic acid solution. The ammonium acetate solution may be made by mixing 80 c. c. of strong ammonia, 100 c. c. of water, and 70 c. c. of acetic acid 99 per cent pure.

Heat the liquid containing the lead vanadate precipitate on the steam bath for 1 hour or more, filter on a tight filter, and wash with warm water. Dissolve the precipitate in the least possible quantity of hot dilute nitric acid, neutralize as before, add 3 c. c. of nitric acid in excess, add 2 c. c. of lead acetate solution and repeat the precipitation of lead vanadate by adding ammonium acetate in excess, filter and add the filtrate to the one from the first precipitation of lead vanadate. Reserve the precipitate of lead vanadate for treatment as described below. Evaporate the united filtrates from the lead vanadate to about 400 c. c. Add 10 c. c. of strong H_2SO_4 to separate the bulk of the lead (derived from the excess of lead acetate) as $PbSO_4$, filter and wash the precipitate with cold water. Neutralize the filtrate from the $PbSO_4$ with ammonia and add freshly prepared $(NH_4)HS$ until the solution is yellow and the uranium and what little lead is present are precipitated as sulphides. Warm the mixture on a steam bath until the sulphides settle well. Filter and wash slightly with warm water.

Dissolve the precipitate in a No. 2 beaker with hot dilute (1:2) HNO_3 add 5 c. c. of H_2SO_4 and evaporate till fumes of H_2SO_4 appear, cool and take up with water, boil, and let the small precipitate of $PbSO_4$ settle until the solution is cold, filter the precipitate and wash it with very dilute H_2SO_4.

SEPARATION OF ALUMINA.

Nearly neutralize the filtrate with ammonia, have the solutions cool (not warmer than 30° C.), and add powdered carbonate of ammonia in about 2 grams excess to precipitate the aluminum. Let the precipitate settle, filter, and wash it with warm water. If the precipitate is bulky or is at all yellow, dissolve it in a little dilute H_2SO_4 and reprecipitate with carbonate of ammonia as described. Acidulate the filtrate from the alumina with H_2SO_4, and boil thoroughly to expel CO_2. Make the liquid slightly alkaline with NH_4OH while it is hot, and heat on the water bath until the ammonium uranate collects and settles. Filter and wash with a very dilute (2 per cent) solution of $(NH_4)NO_3$. Do not allow the precipitate to run dry on the filter after the first washing. Dry the precipitate, ignite it in a porcelain crucible, and weigh as U_3O_8. Dissolve the precipitate in HNO_3 and test it with H_2O_2 for vanadium and with $(NH_4)_2CO_3$ for aluminum.

Dissolve the lead vanadate in dilute HNO_3, add 10 c. c. of H_2SO_4, and evaporate the mixture to fumes. Cool, take up with water, add 10 c. c. of a concentrated solution of SO_2 to the mixture, boil until the excess of SO_2 is expelled, and titrate the hot solution with a standard solution of potassium permanganate. The vanadium compound is reduced by SO_2 from V_2O_5 to V_2O_4. The iron equivalent of the permanganate solution multiplied by $1.6329 =$ the V_2O_5 equivalent. It is not necessary to filter out the lead sulphate before boiling the mixture to expel the SO_2. The boiling is best done in a large flask.

The iron precipitate that was produced by the addition of Na_2CO_3 and H_2O_2 to the original acid solution may contain vanadium. Ignite the precipitate in a platinum crucible and fuse the residue with Na_2CO_3. Leach the fused mass with water, filter, and acidulate the filtrate with H_2SO_4. The filtrate may be added to the main solution before reducing with SO_2, or reduced and titrated separately, as preferred. In expelling the excess of SO_2, it is necessary to boil the liquid for at least 10 minutes after the smell of SO_2 can no longer be detected.

Some of the factors used in calculations are: Fe value of permanganate times $0.9167 = V$. Fe value of permanganate times $1.6329 = V_2O_5$. $1.78124V = V_2O_5$. $U = 0.84824U_3O_8$.

RAPID METHOD FOR THE DETERMINATION OF VANADIUM IN ORES IN THE PRESENCE OF IRON.

Treat 2 to 5 grams of the ore in a 16-ounce Erlenmeyer flask with 20 c. c. of HCl (sp. gr. 1.20) and warm for half an hour. Add 20 c. c. of water and 20 c. c. of H_2SO_4 and evaporate till fumes of H_2SO_4 are liberated. While the mixture is hot add powdered $KMnO_4$, a little at a time, until all organic matter is oxidized and an excess of $KMnO_4$ is present. Heat a few minutes, cool, and add 25 c. c. of water and a few drops of a strong solution of $KMnO_4$ to insure complete oxidation. Add 50 c. c. of HCl (sp. gr. 1.20) and evaporate as rapidly as possible without causing "bumping" until the HCl is expelled and H_2SO_4 fumes are evolved. Continue the fuming for 10 minutes. Cool, add a little cold water, dilute to 250 c. c. with boiling water, and determine the vanadium by titrating the hot solution with a standard solution of $KMnO_4$.

In most cases it is unnecessary to remove the insoluble matter; sometimes, however, it is advisable to do so. Place the ore in a beaker, add 10 c. c. of HCl and 5 c. c. of H_2SO_4, evaporate till fumes of H_2SO_4 are liberated, take up with water, and filter into the flask. Then add 15 c. c. more of H_2SO_4 and proceed as above.

If arsenic and molybdenum are present, they may be removed from the dilute H_2SO_4 solution before filtration by precipitation with H_2S.

Success with this method depends upon complete destruction of the organic matter and complete oxidation of the iron, etc., by $KMnO_4$. Some vanadium ores contain much organic matter. In treating these the H_2SO_4 solution should be heated longer and several cautious additions of powdered $KMnO_4$ should be made. In some cases preliminary calcination at a low temperature will save time.

A glass rod placed in the Erlenmeyer flask lessens the tendency to "bump." The rod should be removed before the acid is completely evaporated.

The method depends on the reduction of vanadic compounds to divanadyl chloride by boiling with HCl and the conversion of this to divanadyl sulphate, while the other metals present remain fully oxidized.

The method of reduction by boiling with HCl was originated by Campagne, who states that reduction is incomplete if conducted in the presence of sulphuric acid, and therefore the acid should not be added until after the vanadium is completely reduced. We have found that in the presence of sufficient ferric iron the reduction by HCl is complete even in the presence of a large excess of H_2SO_4. If the ores under analysis are low in iron and high in vanadium, iron must be added to insure correct results.

MINERALS OF URANIUM AND VANADIUM.

The classifications of the minerals of uranium and vanadium given in most works on mineralogy are incomplete. This incompleteness is due not only to a desire to condense such lists, but also to the fact that the commercial importance of uranium and vanadium compounds is of rather recent growth. For these reasons the authors have compiled from various sources a tabular description of the different minerals of uranium and vanadium. This classification, although necessarily incomplete in some respects, is given below for the use of those who may be interested.

Classification and description of the minerals of uranium and vanadium.

URANIUM MINERALS.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic uranium.	Where found.
			Per cent.	
Autunite	$Ca(UO_2)_2 (PO_4)_2.8H_2O$, uranium and calcium phosphate.	Crystallization, orthorhombic; in plates or tabular crystals; pearly luster, or in micaceous aggregates; translucent; brittle. Cleavage, basal. Color, bright yellow. Hardness, 2 to 2.5. Specific gravity, 3.5 to 3.9.	49 -50.5	Connecticut, Massachusetts, Pennsylvania, North Carolina, South Carolina, Utah, France, Madagascar, Portugal, China, Cornwall, Austria, and Germany.
Blomstrandite	Formula doubtful, tantalo-titano niobate of uranium.	Amorphous		Sweden.
Bröggerite, *see* Uraninite				
Carnotite	$K_2O.2UO_3.V_2O_5.3H_2O$(?), uranyl potassium vanadate(?) containing barium and calcium.	Amorphous, perhaps slightly crystalline; pulverulent, in crusts intimately mixed with quartzose sand. Color, canary yellow. Specific gravity, 4.136.	52 -57	Colorado, Utah, and South Australia.
Chalcolite, *see* Torbernite.				
Clèveite	A variety of uraninite containing uranium, thorium, yttrium, and other rare earths as oxides (rich in helium).	Cubical crystals. Color, velvet black. Specific gravity, 7.49.	48 -61	Norway, Texas, South Carolina, Quebec, and Saxony.
Ebigite, *see* Uranothallite.				Adrianople, Turkey.
Eliasite, *see* Pittinite.	$(PbCaBa)U_3SiO_{12}.6H_2O$ (variety of gummite).			Joachimstahl, Austria.
Euxenite (polycrase).	$R(NbO_3)_2R_3(TiO_3).3H_2O$, titano-niobate of yttrium, erbium, cerium, thorium, and uranium.	Orthorhombic, rarely in crystals, generally in masses. Color, brownish-black; when in powder form, yellowish to reddish brown. Hardness, 6.5. Specific gravity, 4.7 to 5.0.	5 -16	Sweden, Norway, Ireland, North Carolina, and South Carolina
Fergusonite (tyrite, bragite).	$R(NbTa)O_4$, niobate and tantalate of yttrium, erbium, and other rare earths.	Tetragonal. Luster dull, on fractures vitreous and submetallic. Color, brownish-black; brittle. Hardness, 5.5 to 6. Specific gravity, 5.8.	2.1- 8	Sweden, Norway, Greenland, Ceylon, Russia, Massachusetts. North and South Carolina, and Llano County, Tex.
Fritzcheite	$(U, Mn, Va)PO_4.H_2O$	Square tables. Color, reddish-brown.		Bohemia and Saxony.
Gummite	$(PbCaBa)U_3SiO_{12}.6H2O$, alteration product of uraninite, often surrounding a black core of uraninite.	Amorphous, slightly crystalline. Color, reddish-yellow to orange-red and reddish-brown. Specific gravity, 3.9 to 4.2.	55 -60	Saxony, Bohemia, and North Carolina.
Hatchettite	$R(NbTa)_2O_6.H_2O$, tantalo-niobate of uranium.	In octahedrons. Color, yellowish brown. Specific gravity, 4.77 to 4.9.	12.5-13.5	Mitchell County, N. C.
Johannite	Formula doubtful; a hydrous sulphate of uranium and copper.	In druses or reniform masses; monoclinic. Color, apple green.	56	Bohemia and Saxony.
Kochelite, *see* Fergusonite.				
Liebigite	$CaCO_3(UO_3)CO_2.20H_2O$, hydrous carbonate of uranium and calcium.	Transparent; luster vitreous on fracture. Color, apple green; occurs in thin coatings or mammillary concretions in uraninite.	31.5	Austria and Saxony.

Classification and description of the minerals of uranium and vanadium—Continued.

URANIUM MINERALS—Continued.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic uranium.	Where found.
			Per cent.	
Mackintoshite	$UO_2 3ThO_2 3SiO_2.3H_2O$, silicate of uranium, thorium, cerium, and other rare earths. Alteration seems to yield thorogummite.	Tetragonal in square prisms with pyramids; commonly massive, nodular. Fracture small, subconchoidal; luster, dull. Color, black; opaque. Hardness, 5.5. Specific gravity, 4.38 to 5.	20	Llano County, Tex.
Medjidite	Sulphate of uranium and calcium.	Transparent; slightly crystalline.	....	Adrianople, Turkey.
Microlite	Pyro-tantalate of calcium, tungsten, tin, manganese, berylium, yttrium, cerium, and other rare earths.	Regular octahedrons in brown, brownish, and reddish crystals. Hardness, 5.5. Specific gravity, 5.49 to 6.13.	1.6	Virginia, Connecticut, Massachusetts, Greenland, Sweden, and Elba.
Nivenite, *see* Uraninite.				
Nohlite, *see* Samarskite.				
Phosphuranylite	$(UO_2)_3.P_2O_8.6H_2O$, uranyl phosphate	Crystalline, pulverulent, incrustations of deep lemon-yellow color.	60–64	Mitchell County, N. C.
Pilbarite	A hydrated silicate of lead, uranium, and thorium.	Color, bright canary-yellow. Hardness, 2.5 to 3. Specific gravity, 4.4 to 4.7.	....	Pilbarra, Goldfield, Australia in tantalite lode.
Pitchblende, *see* Uraninite.				
Polycrase, *see* Euxenite.	Niobate and titanite of yttrium, cerium, erbium, uranium.	Orthorhombic; prismatic or tabular crystals; luster, vitreous to resinous. Color, black or brownish. Hardness, 5 to 6. Specific gravity, 4.97 to 5.04.	[a]6.3–[b]4.6	Sweden, Norway, and North and South Carolina.
Priorite, *see* Blomstrandite.	$(UO_2,UO_3.ThO_2)$	....	....	Swaziland, Africa.
Randite	Carbonate of uranium?	Yellow incrustations in granite.	....	Pennsylvania.
Rowlandite	Silicate of yttrium	Massive with gadolinite. Color, drab-green.	.3–.5	Llano County, Tex.
Rutherfordin	Uranyl carbonate of lead, etc.	Yellow crystals	69.5	German Africa.
Rutherfordite, variety of fergusonite.	Titanite of cerium	Monoclinic crystals. Color, brownish-black.	68	Rutherford County, N. C.
Samarskite	$R_3R_2(NbTa)_6O_{21}$, tantalo-niobate of yttrium, erbium, calcium, uranium, thorium, iron.	Orthorhombic crystals; massive; fracture, conchoidal; luster, vitreous to resinous. Color, velvety black. Hardness, 5 to 6. Specific gravity, 5.6 to 5.8.	3–14	Sweden, Norway, North Carolina, Colorado, and Canada.
Schroeckinergite	Carbonate of uranium and sulphate of calcium.	Orthorhombic; various yellow colors.	....	Joachimstahl, Austria.
Thorogummite	$UO_2 3ThO_2 3SiO_2.6H_2O$, silicate of uranium, thorium, lead, etc.; occurs with fergusonite, etc.	Color, brownish black	18	Llano County, Tex.
Thorianite	$ThO_2.U_3O_8$, oxide of uranium, thorium, and other rare earths.	Black cubic crystals	4–10	Ceylon (Gambola and Saboragamuva).
Thorite (uranothorite).	$ThSiO_4$, anhydrous silicate of thorium.	Tetragonal and also massive. Hardness, 4.5 to 5. Specific gravity, 4.8 to 5.4.	1–19	New York and Norway.
Torbernite	$Cu(UO_2)_2.P_2O_8.8H_2O$, Copper uranite.	Tetragonal crystals, foliated; micaceous; brittle. Luster, pearly; subadamantine. Transparent to translucent. Color, emerald green and grass-green, some specimens apple or siskin green. Hardness, 2 to 2.5. Specific gravity, 3.4 to 3.6.	47–51.4	Cornwall, Saxony, and Austria.

[a] In crystals.

[b] Massive.

Classification and description of the minerals of uranium and vanadium—Continued.

URANIUM MINERALS—Continued.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic uranium.	Where found.
			Per cent.	
Troegerite	$(UO_2)_3As_2O_8.12H_2O$	Monoclinic crystals in druses of tabular crystals. Color, lemon yellow.	56.2	Saxony.
Uraninite	UO_3UO_2PbO, etc., oxide of uranium with rare earths, Pb, Ca, Fe, Bi, Mn, Cu, Si, Al.	Rarely in octahedral crystals; usually massive. Luster, submetallic, dull. Fracture, conchoidal; brittle. Color, black to dark brown, olive green, grayish; opaque. Hardness, 5.5. Specific gravity, 9 to 9.7 in crystals, or 6.4 and higher when massive.	65–80	Connecticut, North Carolina, Texas, Central City, Colo., South Dakota, Ottawa (Quebec), Saxony, Austria-Hungary, Norway, Spain, and East Africa.
Uranochalcite, *see* Uranopilite.				
Uranocircite	$Ba(UO_2)_2.P_2O_8.8H_2O$	Orthorhombic	47	Germany.
Uranophane	$CaO(UO_3)_2(SiO_2)_2.6H_2O$, Pb, Ba.	In radiated aggregations; massive; fibrous. Color, yellow. Specific gravity, 3.81 to 3.9.	43–56	Silesia, Bavaria, North Carolina, and Spain.
Uranopilite	Uranyl-calcium sulphate	Crystalline. Color, yellow in velvety incrustations.	64	Saxony.
Uranosphærite	$(BiO)_2U_2O_7.3H_2O$	In hairlike globular, and aggregate forms. Color, orange yellow to brick red.	42	Saxony.
Uranospinite	$Ca(UO_2)_2As_2O_8 8H_2O$	In tabular orthorhombic crystals rectangular in outline. Color, siskin green.	49	Saxony and Utah.
Uranothallite (flutherite).	$2CaCO_3U(CO_3)_2.10H_2O$, occurs on pitchblende.	In scaly or granular crystalline aggregations; orthorhombic small crystals. Color, siskin green.	31–32	Austria.
Uranothorite, *see* Thorite.				
Voglianite, uncertain uranium sulphates from Joachimsthal, Austria, *see* Uranopilite.				
Walpurgite	$Bi_{10}(UO_2)_3(OH)_{24}(AsO_4)_4$	Triclinic, in thin yellow crystals resembling gypsum. Specific gravity, 5.76.	16.5	Saxony.
Yttrocasite	Titanate of yttrium, thorium, and uranium (complex).	Color, yellow to black	2.2	Burnet County, Tex.
Yttrotantalite	Tantalate and niobate of yttrium, erbium, iron and uranium, tungsten and tin.	Luster, submetallic to vitreous and greasy. Color, black, brown, brownish yellow, straw yellow; opaque, subtranslucent.	.5–3.7	Sweden.
Xenotime	YPO_4, essentially yttrium phosphate (also cerium, iron, manganese, calcium).	In crystals resembling zircon in habit; in rolled grains; brittle. Luster, resinous to vitreous. Color, yellowish brown, reddish brown, flesh red, grayish white, wine yellow, pale yellow. Hardness, 4 to 5. Specific gravity, 4.45 to 4.56.	2.9	Norway.
Zeunerite	$Cu(UO_2)_2As_2O_8.8H_2O$, uranyl-copper arsenate.	In tabular crystals, tetragonal, resembling torbernite in form and color. Specific gravity, 3.2.	50–53	Saxony, Austria, and Cornwall.
Zippeite, *see* Voglianite.				

The following minerals also contain uranium: Ainnerodite, 13 per cent; davidite (Olari, South Australia); hielmite, 4.1 per cent; naegite (silicate; a mineral found in Japan), 23 per cent; sipylite (complex niobate), 3.3 per cent; uraconite (uranocher, see uranopilite); vetinghofite, see samarskite; voglite (complex carbonate), 30.7 per cent.

Classification and description of the minerals of uranium and vanadium—Continued.

VANADIUM MINERALS.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic vanadium.	Where found.
			Per cent.	
Ardennite (dewalquite).	Vanado-silicate of aluminum, manganese, also containing arsenic.	Prismatic crystals resembling ilvaite. Color, yellow to brownish-yellow. Hardness, 6 to 7. Specific gravity, 3.62.	9	Ardennes, Belgium.
Bismuth ocher		Pulverulent, earthy. Color, yellow, straw color.	1–29	
Brackebuschite (near descloizite).			24–25	Argentina.
Calcium vanadate		Radiated aggregations; pulverulent. Color, claret.		Colorado and Utah.
Calciovolborthite	$(CuCa)_3V_2O_8(CuCa)(OH)_2$, basic vanadate of copper and calcium.	In thin green tables; also gray, finely crystalline, granular.	37–39	Colorado and Thuringia, Germany.
Carnotite	$K_2O.2UO_3V_2O_5.3H_2O$. uranyl-potassium vanadate(?) containing barium and calcium.	Amorphous, perhaps slightly crystaline, pulverulent incrustations; intimately mixed with quartzose sand. Color, canary yellow. Specific gravity, 4.136.	19–21	Colorado, Utah, and South Australia.
Chileite, *see* Mottramite.			13–14	
Cuprodescloizite, *see* Descloizite.	$(PbZnCu)_2(OH)VO_4$ (also called tritochorite, ramirite).		17–22	Colorado.
Dechenite, *see* Descloizite.	PbV_2O_6	Massive; botryoidal; nodular. Color, deep red to yellowish-red and brownish-red. Specific gravity, 5.6 to 5.81.	45–47	Rhenish Bavaria.
Descloizite	$(PbZn)_2(OH)VO_4$, basic vanadate of lead, zinc, etc.	In small crystals; also massive, fibrous, radiated with mammillary surface. Color, cherry-red and brownish-red black. Hardness, 3.5. Specific gravity, 5.9 to 6.2.	20–22	Argentina, Carinthia, New Mexico, and Arizona.
Endlichite	Variety of vanadinite with arsenic.			
Eusynchite, *see* Descloizite.			17–24	
Fritzcheite	Compounds of uranium, manganese, vanadium, phosphorus.	Square tables, reddish-brown.		Autun (France), Bohemia, and Saxony.
Mottramite	$(CuPb)_5V_2O_{10}.2H_2O$, vanadate of lead and copper (possibly identical with psittacinite).	Velvety black incrustations.	17–18	Cheshire, England.
Patronite	Sulphide of vanadium (iron, sulphur, silica, molybdenum, nickel, etc.).	Black to grayish-black	18–25	Minasragra, Peru.
Psittacinite	Formula doubtful, vanadate of lead and copper.	Thin coatings; also pulverulent. Color, siskin to olive green.	17–26	Montana.
Pucherite	$BiVO_4$, bismuth vanadate.	In small orthorhombic crystals. Color, reddish-brown. Hardness, 4. Specific gravity, 6.249.	22–27	Schneeberg, Saxony.
Roscoelite	$H_8K_2(MgFe)(Al,V)_4(SiO_3)_{12}$(?), vanadium mica.	In minute scales; structure micaceous. Color, brown to greenish brown and greenish. Specific gravity, 2.92 to 2.94.	21–29	California, Colorado, and Utah.

Classification and description of the minerals of uranium and vanadium—Continued.

VANADIUM MINERALS—Continued.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic vanadium.	Where found.
			Per cent.	
Vanadic ocher			100	
Vanadinite	$(PbCl)Pb_4(VO_4)_3$, chloro-vanadate of lead.	Hexagonal-pyramidal crystals, prismatic, often hollow prisms, in globules and incrustations; fracture uneven to flat, conchoidal; brittle. Color, deep ruby-red, light-brownish, yellow and reddish-brown; subtranslucent to opaque. Hardness, 2.75 to 3. Specific gravity, 6.66 to 7.10.	8-21	New Mexico, Arizona, Ural (Russia), Carinthia, Sweden, and Argentina.
Vanadiolite	Silico-calcium vanadate.		44-45	
Volborthite	$(CuCaBa)_3(OH)_3VO_4.6H_2O(?)$, hydrous vanadate of copper, barium, and calcium.	In small six-sided tables and in globular form. Color, olive-green, lemon-yellow.	14-15	Ural (Russia), and Colorado.

PUBLICATIONS ON MINERAL TECHNOLOGY AND METHODS OF MINING.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

Bulletin 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

Bulletin 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

Bulletin 44. First national mine-safety demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

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Technical Paper 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

Technical Paper 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig.

Technical Paper 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

Technical Paper 41. Mining and treatment of lead and zinc ores in the Joplin district, Mo., a preliminary report, by Clarence Wright. 1913. 43 pp., 5 figs.

INDEX.

A. Page.

Adams, Orr J., work of 9
Alsdorf, G. W., work of 9
Alumina, separation of, from carnotite ores 89, 90
Annie May claim (Colo.), carnotite deposits at, description of 28
Arnold, J. O., work of 57
Australia, uranium deposits in, description of 50, 51
Austria, pitchblende deposits in, description of 48, 49

B.

Beck, Richard, work of 48
Belcher mine (Colo.), pitchblende deposits in, description of 45
Black Fox claim (Colo.), carnotite deposits at, description of 28, 29
Blackbourne claim (Colo.), carnotite deposits in, description of 27
Bleeker, W. F., process of, for recovery of vanadium, description of 70, 71, 78
Bob-o-link claim (Colo.), carnotite deposits at, description of 29
Bobtail claim (Colo.), carnotite deposits at description of 28
Boltwood, B. B., work of 66, 68
Boot Leg claim (Colo.), carnotite deposits at, description of 29
Boutwell, J. M., work of 14
Brown, H. C., work of 9
Brown, H. L. Y., work of 50
"Bug hole," definition of 20
    description of 30, 31
Bull Canyon, Colo., carnotite deposits near, description of 28, 29

C.

Calhoun mine (Colo.), pitchblende deposits in, description of 46
California, vanadium deposits in, description of 54
Cameron, Angus, work of 9
Carnotite, chemical treatment of, possibility of 41
    concentration of, necessity of 35, 36
        possibility of 35
    deposits of, description of 9–29
        extent of 41, 42
    extraction of radium from 77
    in Austria, extent of 42
    in United States, importance of 42
    mining of, cost of 32
        methods of 32
    occurrence of 36
    origin of 30
    separation of alumina from 89, 90
    transportation of, cost of 33, 34
        difficulties of 33
    treatment of, with nitric or hydrochloric acid 76, 77

Page.

Carnotite ore region, map showing 10
Cassel, Ernest, work of 62
Caywood claims (Colo.), carnotite deposits at, description of 10, 11
Cherrington, C. S., work of 9
Cliff mine (Colo.), carnotite deposits in, description of 25, 26
        view of 24, 26
Cloud claim (Colo.), carnotite deposits at, description of 29
Club Ranch, Colo., carnotite deposits at, description of 24, 25
Coal Creek, Colo., carnotite deposits near, description of 10, 11
Concentration of ores, advantages of 35, 36
    cost of 39, 40
        factors in 39
    dry, methods of 38
        tests of, results of 39
    necessity of 35, 36
    possibility of 35, 40
    wet, method of 36
        tests of, results of 37
Cripple Creek claim (Colo.) carnotite deposits at, description of 23
Cross, —, work of 51
Cumenge, —, work of 18
Cummings, W. L., work of 9
Cummings claims (Colo.), carnotite deposits at, description of 28, 29
Curran, T. V., work of 9
Curran claims (Colo.), carnotite deposits at, description of 23
Cutter, N. Mex., vanadium deposits at, description of 53

D.

Dry concentration. *See* Concentration.

E.

Eagle, Colo., vanadium deposits near, description of 53, 54
East Paradox Valley, Colo., carnotite deposits at, description of 26–28
Electroscope, description of 64
    figure showing 66
    method of using 64
    standardization of, correction for 68
        method of 66, 67
    use of, in determination of ores 64
        precautions necessary in 64
    view of 26
Elkhorn claims (Colo.), carnotite deposits at, description of 10, 11
Engle, W. D., work of 72, 85

F.

Fawn claim (Colo.), carnotite deposits at, description of 29
Fischer, Siegfried, process of, for recovery of vanadium 73

Page.
Fleck, Herman, work of. 9, 19, 51, 74
Fleck, method for extraction of uranium and vanadium, description of. 74
Forsman claim (Utah), carnotite deposits at, description of. 14
French, S. W., work of. 51
Friedel, —., work of. 18

G.

Gale, H. S., work of. 10
German mine (Colo.), pitchblende deposits in, description of. 45
Germany, pitchblende deposits in, description of. 48, 49
Gilpin County, Colo., pitchblende deposits in, description of. 43–46
Grahamite, analysis of. 55
  deposits of. 54
  description of. 54
Grand View claim (Utah), carnotite deposits at, description of. 15
Green River, Utah, carnotite deposits near, description of. 13–16
Greenback claims (Colo.), carnotite deposits at, description of. 27
Greystone claim (Colo.), carnotite deposits at, description of. 28

H.

Haitinger, Ludwig, work of. 79
Haldane, W. G., work of. 19, 51
Hall, Henry, work of. 9
Happy Thought claim (Colo.), carnotite deposits at, description of. 28
Haynes, J. H., work of. 72
Head, George A., work of. 9
Hequembourg, K. D., work of. 9
Hess, F. L., work of. 14, 19, 51, 53
Hillebrand, W. F., work of. 19, 21, 29, 30, 51, 52, 82, 84
Holliday claim (Colo.), carnotite deposits at, description of. 28
Huerfano County, Colo., vanadium deposits in, description of 52, 53
Humery, —, work of. 49
Hydraulic, Colo., carnotite deposits at, description of. 29

I.

Iveagh, —, work of. 62

J.

Jacobs claim (Colo.), carnotite deposits at, description of. 26
Jasper claim (Colo.), carnotite deposits at, description of. 28
Jo Dandy claim (Colo.), carnotite deposits in, description of. 27

K.

Kent Smith claim (Colo.), carnotite deposits at, description of. 27
Kimball, Gordon, work of. 18
Kirk mine (Colo.), pitchblende deposits in, description of. 44
Koenig, G. A., process of, for recovery of vanadium. 73
Kuppelweiser, —, work of. 61

L. Page.

Last Chance claim (Colo.), carnotite deposits at, description of. 28
Little Emma claim (Colo.), carnotite deposits at, description of. 12
Little Ruth claim (Utah), carnotite deposits at, description of. 15
Little Tom claim (Colo.), carnotite deposits at, description of. 28
Logan, Thomas, work of. 18
Long Park, Colo., carnotite deposits at, analyses of. 20
  description of. 21–24
Lookout claim (Colo.), carnotite deposits at, description of. 12
Lorimer claim (Utah), carnotite deposits at, description of. 14, 15
Loveless-Forsman claim (Utah), carnotite deposits at, description of. 15

M.

McBride, James, work of. 18
McIntyre district, Colo., carnotite deposits at, description of. 29
McKeever claim (Colo.), carnotite deposits at, description of. 26
Meeker, Colo., carnotite deposits near, description of. 10, 11
Melrose Discovery claim (Utah), carnotite deposits at, description of. 15
Mitchell County, N. C., pitchblende deposits in, description of 43
Monayunk claim (Utah), carnotite deposits at, description of. 15
Monogram claim (Colo.), carnotite deposits at, description of. 27

N.

Napoleon claim (Utah), carnotite deposits at, description of. 15
Nevada, grahamite deposits in, description of. 54
Newmire, Colo., vanadium deposits near, description of. 51, 52
Nitric acid, use of, in treatment of ores. 76, 77
North Star mine (Colo.), carnotite deposits in, view of. 16

O.

Oklahoma, grahamite deposits in, description of. 54
Opera Box claim (Colo.), carnotite deposits at, description of. 27

P.

Paradox Valley, Colo., carnotite deposits at, description of. 19–29
  map of. 22
Patronite, deposits of, in Peru. 55
  description of. 55, 56, 95
Pitchblende, concentration of, possibility of. . 47
  deposits of, description of. 43–47
  method of treating. 79, 80
  wastes in mining of. 46, 47
Placerville, Colo., vanadium deposits near, description of. 51, 52
Portugal, uranium deposits in, description of. 49, 50
Poulot, —, work of. 18
Purrington, —, work of. 51

Q. Page.

Quarter Circle claim (Colo.), carnotite deposits at, description of ... 27

R.

Radcliff, Sidney, method of, for recovery of radium ... 75
Radioactivity of sandstone. *See* Sandstone, radioactivity of.
Radioactivity of water, variations in ... 59
Radium, determination of ... 66
        method for ... 68, 69
    extraction of, from carnotite ore ... 77
        from pitchblende ... 79, 80
        importance of ... 70
        method of ... 75
    misuse of, in treatment of disease ... 61
    price of ... 62
    production of, in foreign countries ... 57
        in United States ... 56
    sale of, fraud practiced in ... 62
    use of, in treatment of disease ... 58, 59, 63
            apparatus for ... 59
            results of ... 60
Radium Institute, English, work of ... 62
Radium Institute of Vienna, scope of ... 61
Ransome, F. L., work of ... 19, 21, 29, 30, 51, 52
Richardson, Clifford, work of ... 54
Rickard, Forbes, work of ... 9, 45
Rutherford, E., work of ... 66

S.

San Miguel County, Colo., vanadium deposits in, description of ... 51, 52
San Raphael River, Utah, carnotite deposits near, description of ... 14
Sandstone, radioactivity of ... 31
Saucer Basin, Colo., carnotite deposits at, description of ... 25, 26
Saunders claim (Colo.), carnotite deposits at, descriptions of ... 29
Segaud, —, work of ... 49
Skull Creek Basin, Colo., carnotite deposits in, description of ... 11, 12
Spencer, —, work of ... 51
Stewart, Newt., work of ... 9
Sundown claim (Colo.), carnotite deposits at, description of ... 29
Swindler mine (Colo.), carnotite deposits at, description of ... 23
    view of ... 24

T.

Table Mountain, Utah, carnotite deposits at, description of ... 16
Taff, J. A., work of ... 54
Talbert, Andrew J., work of ... 18
Taylor, David, work of ... 9
Telluride No. 8 claim (Utah), carnotite deposits at, description of ... 17
Thompsons district, Utah, carnotite deposits at, description of ... 16–18
        view of ... 16

Page.

Thunderbolt mine (Colo.), carnotite deposits in, description of ... 26
Turner claims (Colo.), carnotite deposits at, description of ... 21–23

U.

Ulrich, Karl, work of ... 79
Uraninite, separation of emanation from, apparatus for, figure showing ... 67
Uranium, deposits of, in Australia, description of ... 50, 51
        in Portugal, description of ... 49, 50
    determination of, difficulties in making ... 82
        methods for ... 82–86, 88, 89
    origin of ... 30
    percentage of, in ore ... 34
    recovery of, method of ... 72
    uses of ... 58
Uranium minerals, list of ... 92–95
Uranium oxide, percentage of, method of calculating ... 65
    production of ... 56
    selling price of ... 34, 35

V.

Valley View claim (Colo.), carnotite deposits at, description of ... 27
Vanadic acid, extraction of, from copper vanadate ... 78
Vanadium, determination of, difficulties in making ... 82
        method for ... 82–84, 88, 90, 91
    effect of, on steel ... 57
    origin of ... 30
    percentage of, in ore ... 34
    production of, in United States ... 57
    recovery of, method of ... 70–74
    separation of, from uranium ... 78
    uses of ... 57, 58
Vanadium minerals, list of ... 92, 95
Vanadium oxide, selling price of ... 34, 35
Vernon Junior claim (Utah), carnotite deposits at, description of ... 15
Voilleque, ——, work of ... 18

W.

Wardvern claim (Utah), carnotite deposits at, description of ... 15
Wedding Bell claim (Colo.), carnotite deposits at, description of ... 29
West Virginia, grahamite deposits in, description of ... 54
Wet concentration. *See* Concentration.
White, I. C., work of ... 54
Whittemore, Charles F., work of ... 9
Widow claim (Colo.), carnotite deposits at, description of ... 29
Willmarth, O. B., work of ... 9
Wilson claims (Colo.), carnotite deposits at, description of ... 25, 26, 28
Wood mine (Colo.), pitchblende deposits in, description of ... 46